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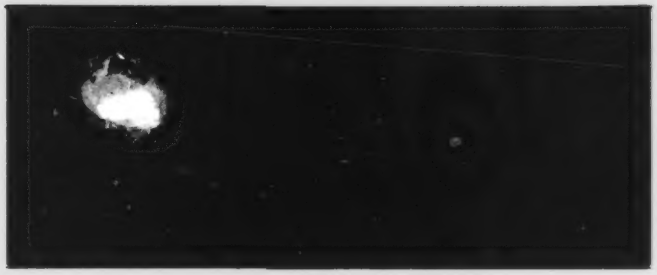

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CONTRACT SERIAL NUMBER 69108-01-SQ
CONTRACT REQUISITION NUMBER 6-9108
CONTRACTOR: MONENCO CONSULTANTS LTD.

INTEGRATED RETORTING COMBUSTION SYSTEM USING FIXED BED TECHNOLOGY

DSS File No.: 09SQ.23440-6-9108

Scientific Authority: Marten Ternan
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Final Report Submitted

to

MICROMEDIA

MONENCO CONSULTANTS LTD.
St. Catherines, Ontario

108 pp

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Dear Dr. Ternan:

Integrated Retorting Combustion System
Using Fixed Bed Technology

We enclose five copies of the final report prepared by RPC, along with Monenco's summation evaluating the results of the test.

We trust you will find that the work has been brought to a satisfactory and successful conclusion.

Cordially,

A. Ismail

A. Ismail, P. Eng.
Head of Advanced
Energy Technology

AI/st

Enclosures

cc: Dr. P. Salib
Dr. D. Abbott

SUMMATION BY MONENCO

Project: To carry out a single experiment of 48 hours duration, testing an integrated retorting (pyrolysis) combustion system using fixed bed technology.

Objective: To determine whether results obtained under one set of experimental conditions indicate that further testing, under a planned set of variable conditions, is warranted for the purpose of investigating the techno-economic feasibility of a scaled-up process.

It was also intended to evaluate whether a particular contractor, and the experimental setup at the contractor's laboratories, are capable of carrying out such testing.

Evaluation of Results: It is Monenco's view that RPC have satisfactorily completed the contract work and brought it to a successful conclusion; and that the results warrant further testing and development at RPC's laboratory to determine the commercial prospects of an integrated pyrolysis combustion process.

In the context of today's energy scenario, the extraction of a reasonable grade of oil from shale for processing into transportation fuels are goals worth pursuing. Monenco agree with the conclusions and recommendations of RPC's report. It is clear from the report that considerations relating to scaling up, and the economics of utilizing the net energy output of the integrated system, need to be reviewed on the basis of process optimization in terms of the variable parameters, equipment design and system configuration. The test conducted under this program has served the purpose of indicating the directions in which equipment design and system configuration should move. It also lays the basis for determining how the variable parameters should be set in order to obtain meaningful results from a series of planned tests. While the failure of the electrical heater, which provides auxiliary heat to the recirculated retort gas, was accidental, it was also, in a sense, fortuitous in that the operation of the system at a lower heat input has provided an insight with regard to computing system performance and examining the relationship between heat input and retort performance. On the whole, the completion of this test provides a good data base for the next phase of the integrated system program.

The experimental retort/combustor unit at RPC's laboratories can easily be modified for further testing to accommodate those changes in equipment and systems design/configuration which are indicated as desirable by the experiment which has been conducted.

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SUMMARY

This is the final report for DSS Contract No.: 23440-6-9108/01SQ the requirement being the evaluation of "An Integrated Retorting-Combustion System Using Fixed Bed Technology". The contract was awarded to Monenco, with the Research and Productivity Council (RPC) as a subcontractor.

The primary objective of the study was to carry out an extended run for 48 hours duration using RPC's Integrated System, which consists of a circulating fluidized bed combustor and a fixed bed retort.

The circulating fluidized bed combustor system has operated trouble free for the entire duration of the 57 hours of the integrated combustion/retorting run. Minor operational problems were encountered with the retort. Optimum operating conditions were sustained for 20 hours of continuous operation, during which the oil recovery was 97.4% of Fischer assay. The quality of oil compared favourably with that obtained in previous work carried out at RPC. The combustion efficiency was over 90% which is good for a pilot scale unit. Sulphur capture was low because of the low Ca/S molar ratio. A much higher spent shale to Minto coal ratio will be needed to further reduce the sulphur emission.

1.0 INTRODUCTION

The work described in this report was in fulfillment of DSS Contract No.: 23440-6-9108/01SQ dated December, 1986, which has been issued in response to the Research and Productivity Council's Proposal No. AS/87/11P submitted through Monenco Consultants Ltd. The title of the Project is "An Integrated Retorting-Combustion System Using Fixed-Bed Technology". The contract was awarded to MONENCO Consultants Ltd and RPC carried out the work as a subcontractor to Monenco.

The concept of integrating a fluidized bed coal combustor and an oil shale retort system is illustrated in Figure 1. In this system the pyrolysis of oil shale known as retorting is carried out in a fixed bed reactor to produce liquid and gaseous fuels and a by-product spent shale. The heat of retorting is supplied by the flue gases from a circulating fluidized bed combustor operating on a mixture of spent shale and a high sulphur coal. The spent shale which has calcium and magnesium oxides acts as a sulphur capture media. Excess heat from the combustor would be used for steam raising and power generation. The integrated system, therefore, not only utilizes the kerogen and carbonate content in N.B. oil shale but also the residual carbon in the spent shale. All energy is thus recovered from the oil shale and a means of sulphur reduction is provided during the combustion of Minto coal.

The experimental work described in this report has been carried out according to the work statement of the contract as given in Appendix A. To simplify following the report Appendix B. part of the work statement, is a guide to where data can be found in the report.

In summary, this project involved procurement of Minto coal and Albert county oil shale, checking that the RPC pilot plant conformed with the contractor's requirements,

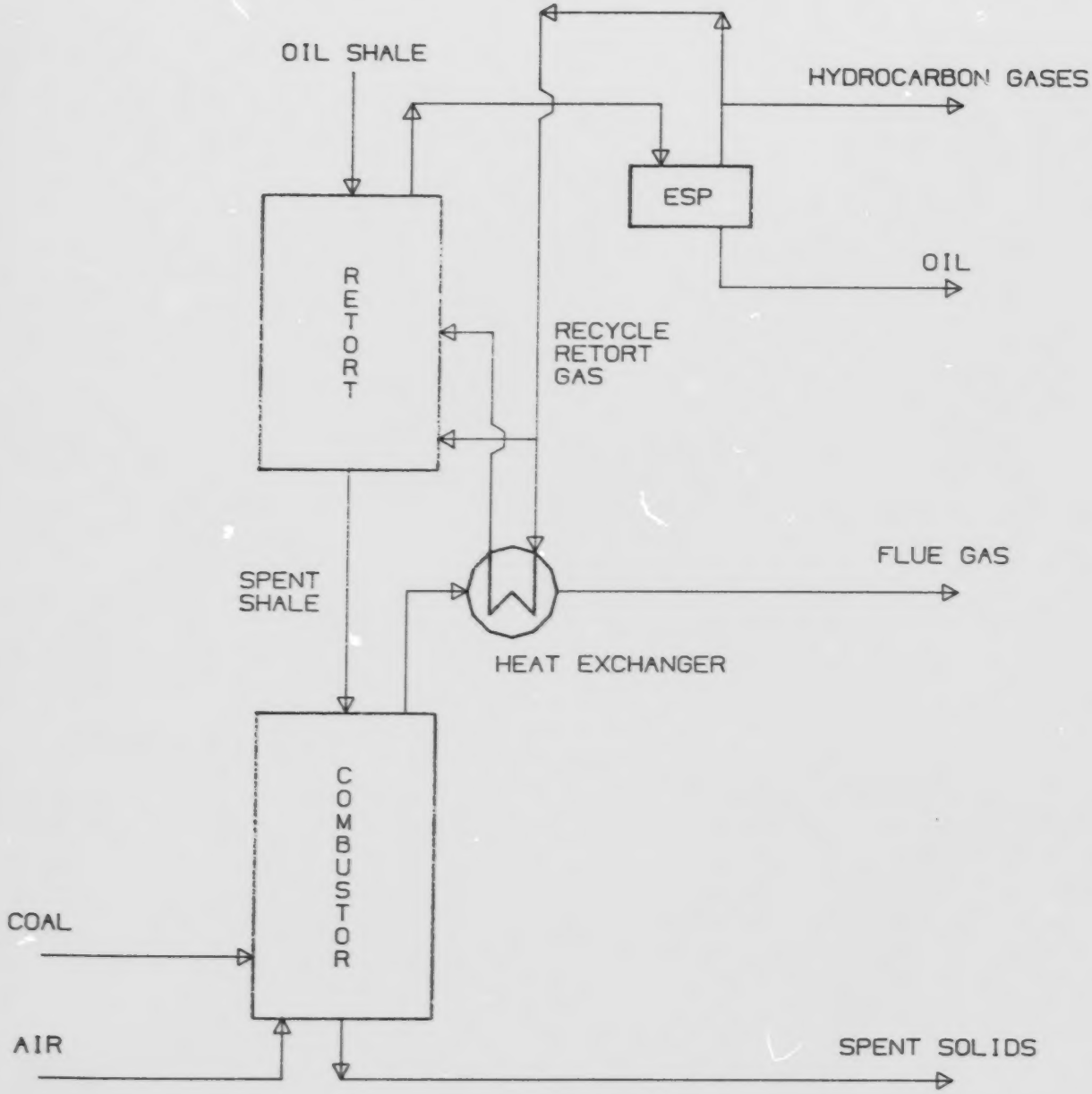

FIG 1 : BLOCK DIAGRAM FOR AN INTEGRATED OIL SHALE RETORT AND COAL COMBUSTOR

making modifications if necessary and setting the operating conditions for a test run. The unit was then to be started up and operated in the integrated mode for 48 hours in steady state condition. Measurements were to be taken regularly in order to evaluate the performance and for the calculation of material and heat balances.

This report gives details of the experimental work, results, discussions and recommendations for future work.

2.0 DESCRIPTION OF RPC'S INTEGRATED SYSTEM

The integrated system consists of three major sub-systems: a circulating fluidized bed combustor, a fixed bed retort and a heat exchanger. A diagram of the integrated system is shown in Figure 2. A description of each sub-system is given below. Unfamiliar abbreviations used in the diagrams and tables are explained in Appendix H.

2.1 Circulating Fluidized Bed Combustor (CFBC) System

The CFBC consists of a combustor, standpipe, primary and secondary cyclones, J-valve, baghouse and other peripheral equipment:

The Combustor

The combustor is cylindrical in shape with a total height of 5.8 m (19') and it has an inside and outside diameter of 12.7 cm (5") and 50.8 cm (20") respectively. The wall consists of three layers of material: an inner layer made up of high temperature refractory, a middle layer of castable insulating refractory, and a carbon steel shell. The thickness of the high temperature refractory and insulating are 5 cm and 13 cm, respectively. A 0.5. million Btu propane burner is used for start-up purposes.

The Stand Pipe

The stand pipe is used for storage and recirculation of solids received from the primary cyclone. It is 2.7 m (9') high and has an inside and outside diameter of 22.8 cm (9") and 61 cm (24") respectively. The stand pipe is also refractory lined and it is also equipped with discharge and sampling ports.

A cooling section is installed at the bottom of the stand pipe. This is used to extract the extra heat generated

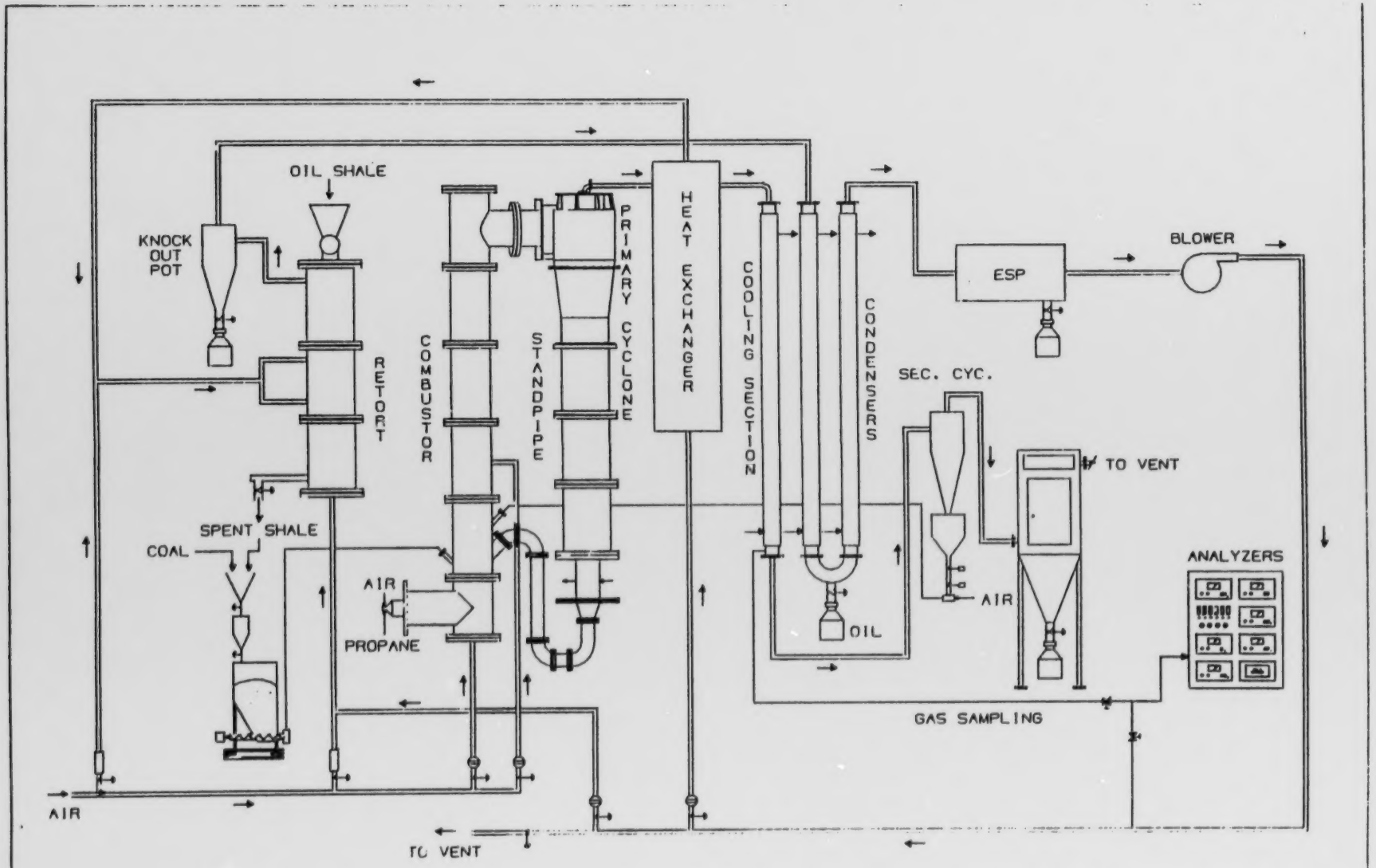

FIGURE 2 : SCHEMATIC OF RPC INTEGRATED SYSTEM

in the combustor so that both bed temperature and air/fuel ratio can be set and maintained at the appropriate levels.

Cyclones

The primary cyclone which is 30 cm (12") in diameter is refractory lined in a similar manner to the reactor and the standpipe. The secondary cyclone is made of stainless steel and is 25 cm (10") in diameter. Solids collected by the secondary cyclone can be recycled pneumatically to the combustor.

J-Valve

Circulation of solids from the standpipe to the reactor is achieved by a non-mechanical J-valve. Air is supplied at two points - one at the base of the standpipe, and the other is at the base of the vertical section of the J-valve. The J-valve is made of 2", (Schedule 40) SS pipe.

Gas Cooling System

Hot flue gases from the combustor are fed into a specially designed heat exchanger described in section 2.3 (See Figures 2 and 3). The gases are then cooled further in water jacketed pipes which simulates the convective section of a boiler. The cooling is carried out in a water cooled one-pass system consisting of two concentric pipes of a nominal diameter of 15 cm (6") (Schedule 40) and 20 cm (8") (Schedule 60), respectively. The height of the pipes is 5.2 m (17').

Baghouse

The flue gases from the cooling section are fed through the secondary cyclone into the baghouse. The latter is a continuous duty self-cleaning dust collector which consist of 16 high temperature bags capable of handling 200 SCFM.

Solid Feeding to the Combustor

The solids, discharged by a screw feeder from a storage bin, are entrained by air and fed to the combustor. The storage bin is placed on a load cell to provide a record of the solid feed rate.

2.2 The Retorting Sub-System

The system consists of a cylindrical retort, solid feeding and withdrawal equipment, product oil recovery system, and gas recycling network. System components are described below:

Retort

The cylindrical retort has an inside and outside diameter of 25.4 cm and 61 cm respectively and a total height of 2.9 m. Like the combustor, the wall consists of three layers of material: an inner layer of castable high temperature refractory, a middle layer of castable insulating refractory and a carbon steel shell. The thickness of the high temperature and insulating refractory are 11.7 cm and 5.1 cm respectively.

The cleaned retort off-gases are recycled to the retort as bottom and middle recycle gas. The middle recycle gas, after spliting into two streams, is introduced at a height of 0.8 and 1.25 m measured from the bottom of the retort.

The oil mist-ladden off gases are withdrawn near the top of the retort through a side port.

Product Oil Recovery System

Off-gas containing oil mist from the retort undergoes three stages of separation for product oil recovery. These are:

(a) a hot knock-out for removing heavy oils and entrained fine particles.

(b) two condenser columns with special packings, with an internal column diameter of 0.15 m and an effective height of 4.7 m.

(c) an electrostatic precipitator (0.55m wide x 1.52m long x 0.13 m thick) with five sets of capacitor cells and ionizers.

Gas Recycling Network

The cleaned off-gases from the oil recovery system are metered and recycled back to the retort. A portion can be withdrawn as a product gas. The middle recycle can be preheated in the heat exchanger prior to injection to the two levels of gas feed points in the retort. At each level, the preheated gas is distributed using two one inch 316 SS pipes placed at 90^{o} with respect to each other. Each pipe contains 10 holes of 2.4 mm diameter.

Solid Feeding and Withdrawal System

Oil shale is fed into the retort by gravity from an overhead hopper through a rotary valve. A predetermined amount of oil shale is loaded into the bin every hour and a timer is used to control the ON/OFF cycles of the rotary valve in order to maintain a steady supply of fresh shale.

Spent shale is discharged from a bin attached to the bottom of the retort through a variable-speed screw feeder into a receiving pot. The discharge rate is set so as to maintain a steady shale bed height in the retort. Bed level is measured periodically with a graduated rod.

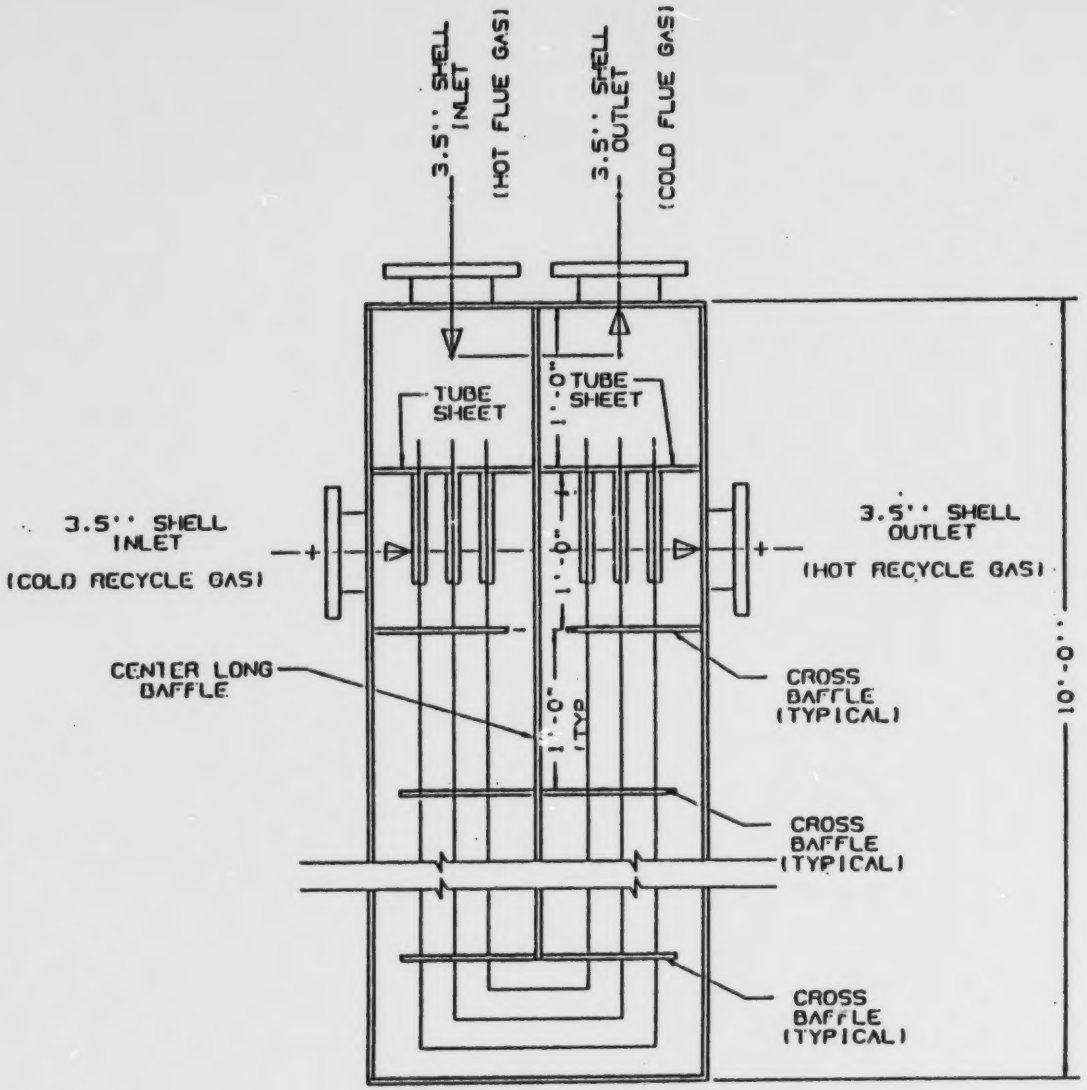

FIG 3 SCHEMATIC OF THE HEAT EXCHANGER FOR THE INTEGRATED SYSTEM

2.3 The Heat Exchanger

The heat exchange between the combustor and the retort gases is carried out in a specially designed shell and tube heat exchanger. It has eleven U-shaped tubes of a total surface area of $4.2m^2$. The schematic of the heat exchanger is shown in Figure 3.

2.4 Gas Sampling System

On-line gas analyzers (hydrocarbon, CO, CO_2, O_2, NO_x, SO_2) are used to monitor the composition of the retort and combustion gases. Retort gas samples are withdrawn from the recycle line and conditioned via a series of filters prior to entering the analyzers. Combustion flue gas samples are withdrawn through a heated line and cleaned by a series of filters. One stream, either combustion flue gas or retort off gas, is continuously monitored, while the other is analyzed as necessary. In addition, tubes of retort gas sample are also collected for further analysis of H_2, N_2 and hydrocarbon content using gas chromatography.

3.0 FEEDSTOCK PREPARATION AND CHARACTERIZATION

As-received crushed oil shale was dried and screened through 5/8" and 1/8" to remove oversize and undersize fraction. As-received coal was first dried followed by crushing and screening. The size analysis, chemical analysis and the Fischer assay analysis of oil shale, and coal are given in Tables 1 and 2.

TABLE 1
Size Analysis of N.B. Oil Shale
and Minto Coal

Screen Size, mm	Weight %
Oil Shale	
+13.5	8.0
+8.0	27.7
+2.83	54.9
+1.41	7.1
-1.41	2.3
Minto Coal	
+2.0	36.1
+0.84	26.4
+0.35	16.4
+0.18	11.9
-0.18	9.2

TABLE 2

Analysis of Coal and Oil Shale

	N.B. Oil Shale	Minto Coal
Proximate Analysis (wt%)		
Moisture	0.98	0.45
Ash	78.98	14.12
Volatile Matter	18.43	34.81
Fixed Carbon	1.61	50.62
Ultimate Analysis (wt%)		
Carbon - Organic	11.90	70.78
Inorganic	1.80	-
Hydrogen	2.30	4.72
Nitrogen	0.45	0.94
Oxygen	3.56	3.41
Sulphur	0.03	5.49
Ash	78.98	14.12
Ash Analysis (wt%)		
SiO_2	59.83	32.22
Al_2O_3	16.88	16.42
Fe_2O_3	7.40	49.58
CaO	5.18	0.32
MgO	1.90	0.27
Others (Na_2O, K_2O etc.)	8.81	1.19
Heating Value		
Btu/lb	2398	12993
J/kg	5.575×10^6	30.2×10^6
Fischer Assay (wt%)		
% Oil Yield	7.4	-
$\%H_2O$ Yield	3.5	-
% Gas Lost	1.1	-
Oil Yield Imp. gal/ton (short)	16.9	-
Oil Density g/ml @ 15.5°C	0.8738	-

4.0 START-UP PROCEDURE

Preheating of the retort and CFBC was initiated simultaneously. The retort reached operational temperature after six hours, while the CFBC required approximately twelve hours before coal feeding could be initiated.

4.1 Retort Start-Up

Retort preheating was with a propane burner placed through a side port located close to the bottom of the retort. When the temperature at the top of the retort was approximately 800°C, the burner was removed and the side port was sealed. The retort was immediately filled with oil shale and operated in the direct retorting mode i.e. air was fed to the retort in order to establish a combustion zone to supply the heat requirements of the retort. The product gases were passed through the cleaning train and recycled back to the retort.

4.2 Combustor Start-Up

The standpipe was filled with 200 kg of sand prior to preheating of the combustor with a 0.5 million Btu/hr propane burner. When the temperature of the combustor was over 700°C, recycling of the sand from the standpipe was initiated. The standpipe was preheated by the hot recycling sand. Coal feeding was started when the standpipe temperature was over 600°C. Preheating of the combustor system took over twelve hours and steady state combustor operation was achieved in approximately 16 hours.

At steady state combustor feed was switched to the coal/spent shale mixture feed (ratio 1:1.5). Coal was 40% of feed mixture.

5.0 EXPERIMENTAL PROCEDURE AND MEASUREMENTS

Once the two systems had been started up, process integration was achieved over a period of 20 hours by preheating the retort gases in the heat exchanger and gradually reducing the air supply to the retort. The spent shale from the retort was crushed, screened to a top size of 1/8", blended with coal and fed into the combustor.

Once steady state operation had been achieved the following readings and measurements were taken every hour:

- feed and product rates of coal, oil shale and spent shale.

- all gas flow rates to both CFBC and the retort

- middle recycle, bottom recycle and exhaust retort gas

- temperature of the combustor, standpipe, retort and at other strategic locations to monitor the process performance

- flue gas and retort gas compositions

- discharge from the standpipe, secondary cyclone, column which was (recycled) and the baghouse

- spent shale, knock-out pot discharge, condensers oil and water ESP oil and water.

Figures 4 to 7 give complete identification of all process streams and show the location of all temperature and flow measurement sensors. The integrated system is shown in Figure 4, while details for the combustor and the retort are shown in Figures 5 and 6, respectively. Figure 7 shows solid sampling location for the combustor.

Samples of the following streams were taken every six hours:

- spent shale, Figure 6
- shale oil, Figure 6
- standpipe, baghouse, secondary cyclone and column discharge, Figure 7
- knock-out pot solids, Figure 6

The experimental measurements required for the project, listed in the work statement, are given in Appendix B together with the tables where the information can be found as well as the number of the figure pertaining to the tests.

6.0 OPERATING EXPERIENCE

Upon completion of the start-up procedure which was sixteen hours in duration, the system was then operated for approximately 20 hours in the integrated mode. During this 20 hour period, the operating parameters were varied within the ranges given in Table 3 which was approved by the Scientific Authority. The system was then operated continuously for 57 hours. The hourly recorded data is given in Appendix C. (Tables 18a-c, 19a-c).

During the 57 hour run all the major components of the system operated reliably, there was however, two small distrubances. At the beginning of the 14th hour of operation, the retort discharge screw seized for a short period. The system was back to normal operation 2 hours later. The second disturbance accurred at the 35th hour. At that time the electrical coil used to boost the recycle gas temperature burnt out resulting in a total loss of supplementary electrical heat to the mid retort recycle gases, see Table 19. Apart from these two minor disturbances the system was maintained at the same set of operating conditions throughout the 57 hour period. The loss of electrical heat however, resulted in lower oil recovery in the last twenty two hours of operation.

For data analysis purposes the 57 hour run will be divided into three periods based on the systems performance. The "initial period" will be is thirteen hours in duration (hours 1 to 13). The 14th and 15th hours will not be considered. The second period will be designated as the "steady state period" which will be 20 hours in duration (15th to 35th hours). During this period the entire system ran smoothly throughout, resulting in high oil recovery. The last period is designated as the "Lower Heat Input" period and it was twenty two hours in duration.

TABLE 3

Integrated Retorting Combustion

System Approved Ranges of Operating Conditions

Retort Operating Conditions

- shale feed rate, kg/hr	P1	18 + 10%
- particle size, inch		0.5 to .125
- gas recycle rate, kg/kg of shale	FI-3	2.25 + 10%
- gas distribution: middle recycle, %		80 + 10%
- bottom recycle, %		20 + 10%
- bed height, m		2 + 10%
- air feed rate to kg/kg of shale the bottom of the retort	FI-2	0 to 0.25

Combustor Operating Conditions

- feed composition (coal/spent shale ratio)		1:1.5
- solids feed rate, kg/hr	C1	16 + 10%
- maximum feed size, inch		0.2 + 10%
- air feed rate, kg/hr	FI-1	100 + 10%
- bed temperatuare, ^{o}C	T20-T23	850 + 5%
- excess air*, %		15 + 10%

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* excess air is not an independent variable

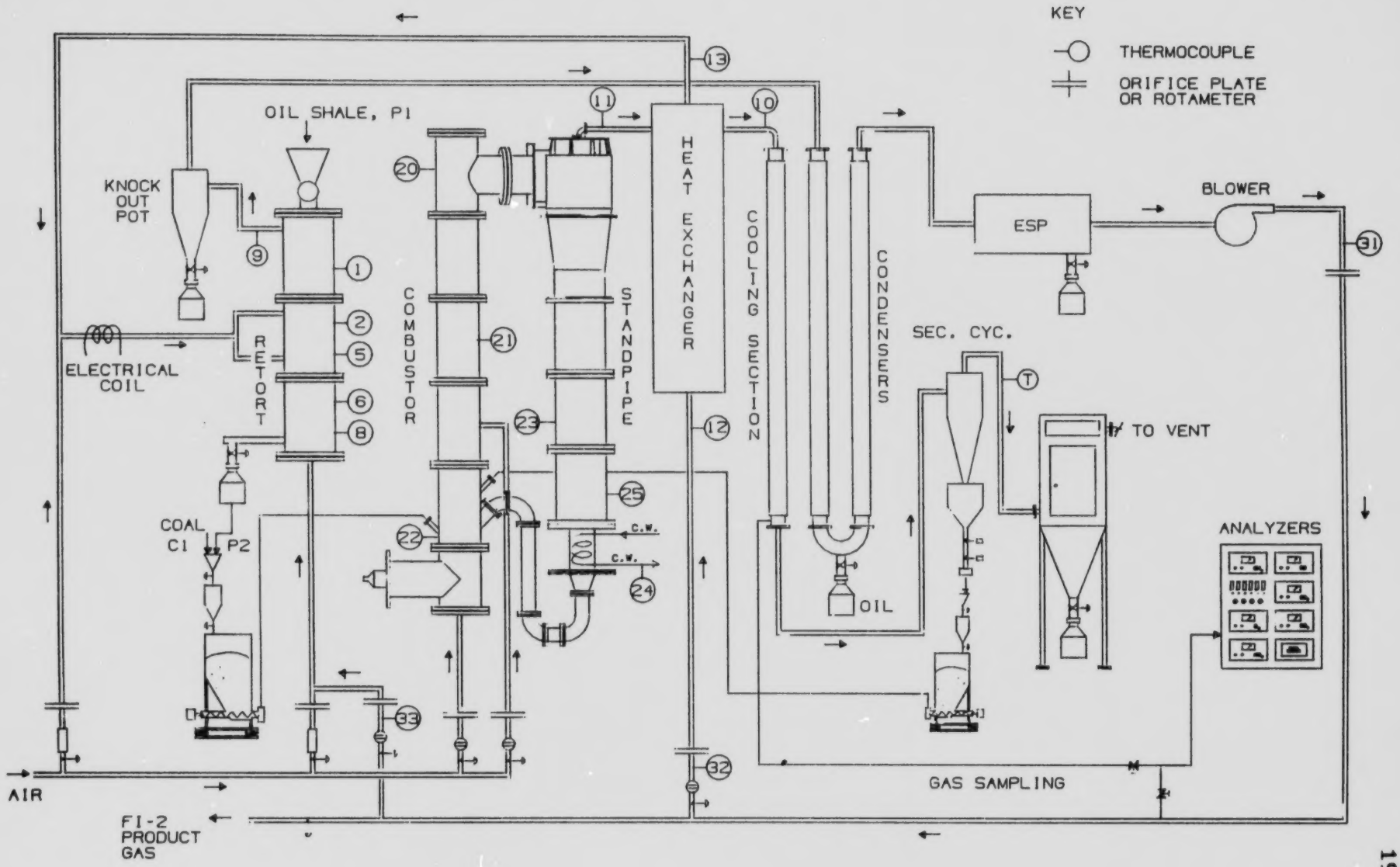

FIGURE 4 : SCHEMATIC OF RPC INTEGRATED SYSTEM SHOWING LOCATION OF INSTRUMENTATION FOR TEMPERATURE AND FLOW MEASUREMENTS

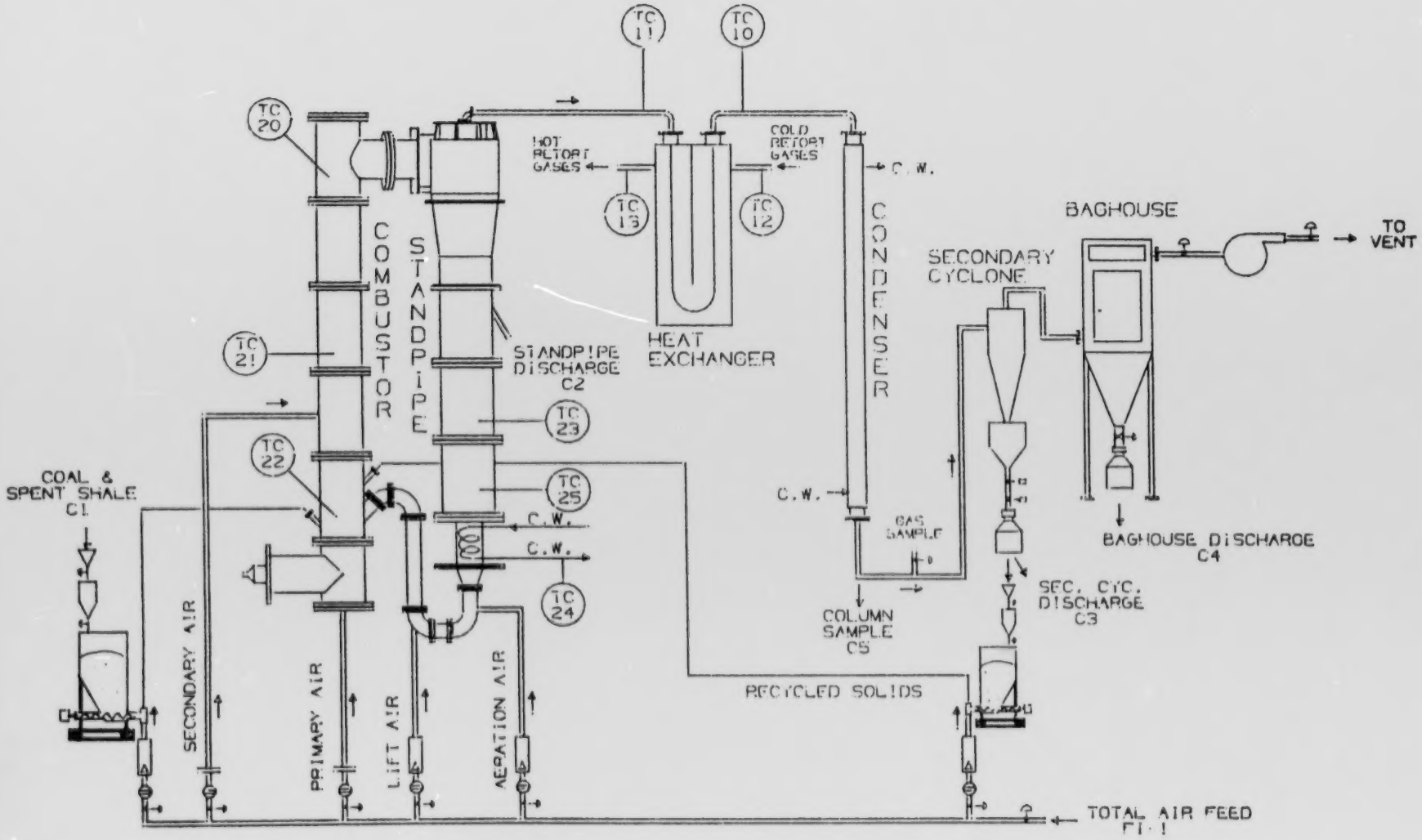

FIGURE 5 : SCHEMATIC OF RPC CIRCULATING FLUIDIZED BED COMBUSTOR SHOWING STREAM SYMBOLS AND INSTRUMENTS FOR TEMPERATURE AND FLOW MEASUREMENTS

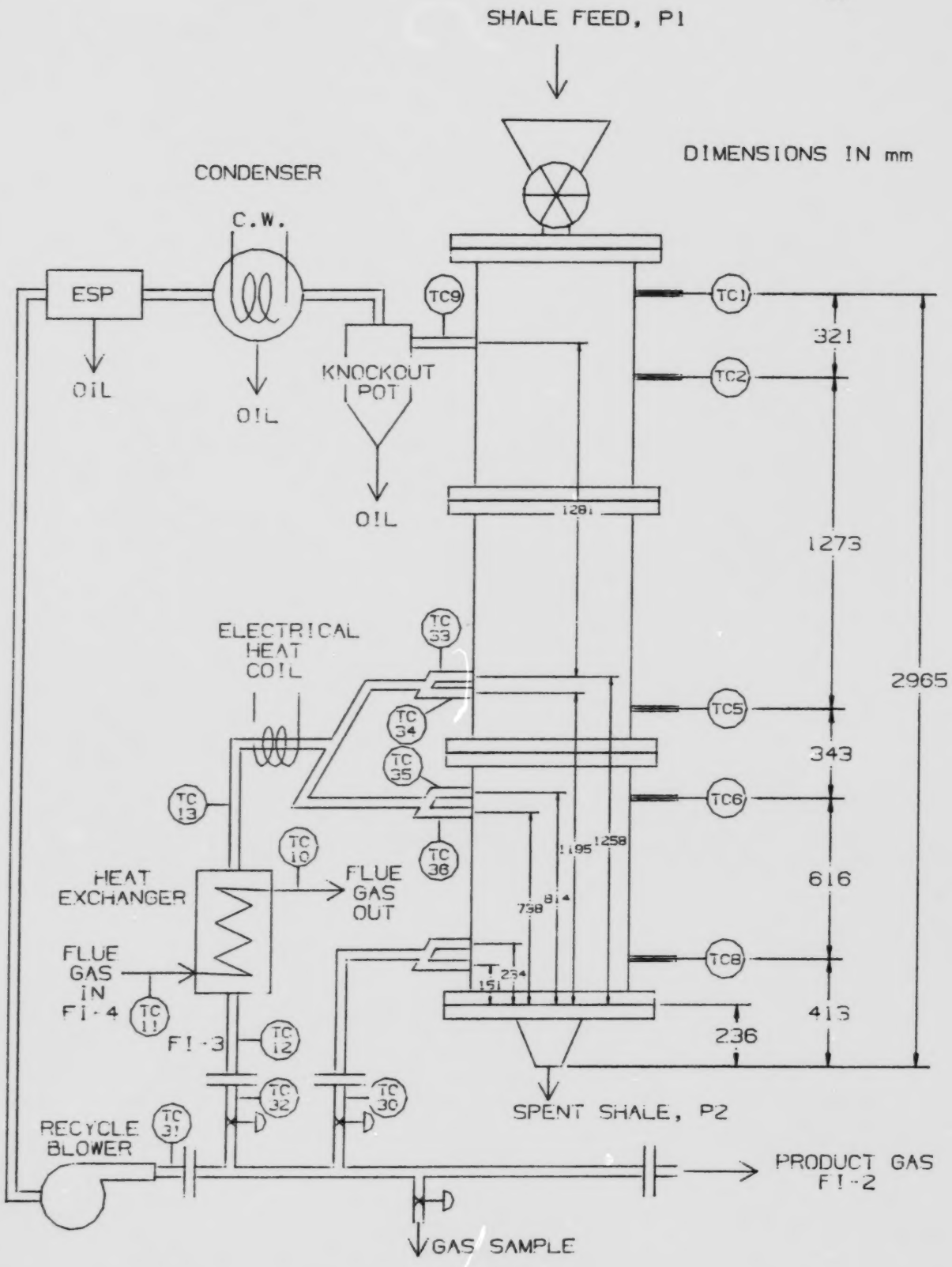

FIG 6 : SCHEMATIC OF RPC FIXED BED RETORT SHOWING STREAM SYMBOLES AND INSTRUMENTATION FOR TEMPERATURE AND FLOW MEASUREMENTS

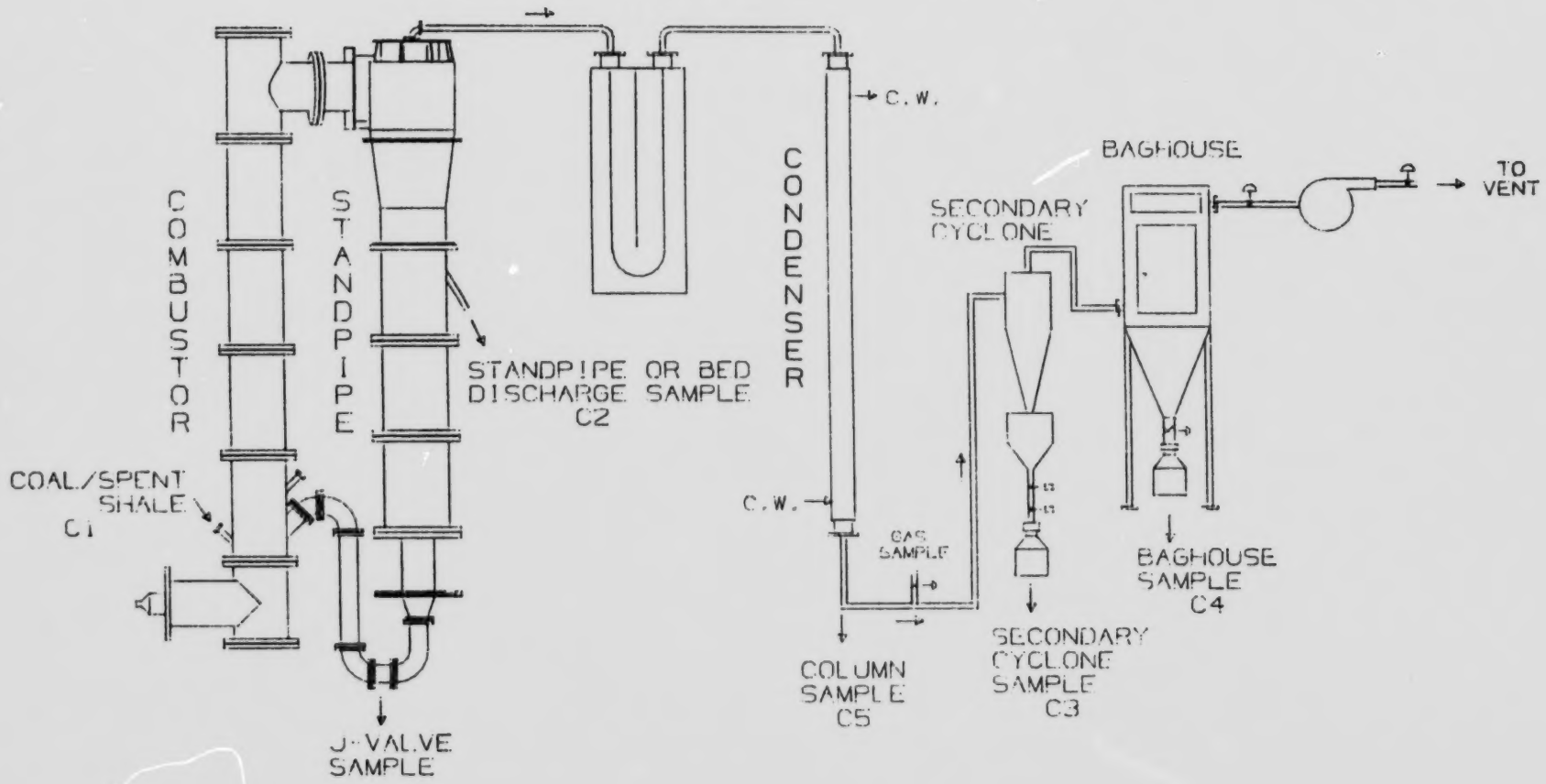

FIGURE 7 : SCHEMATIC OF RPC CIRCULATING FLUIDIZED BED COMBUSTOR SHOWING SOLID SAMPLING LOCATIONS

7.0 RESULTS

7.1 Circulating Fluidized Bed Combustor (CFBC) System

The summary of the CFBC experimental condition and results are shown in Table 4. This data is an average of hourly readings, which are attached in Appendix C.

The excess air level was calculated based on the air required and air supplied. The combustion efficiency was calculated on the basis of carbon input to the system and the carbon discharge from the baghouse, secondary cyclone and the standpipe. The column discharge was recycled to the CFBC.

Mass and heat balance calculations are shown in Table 5. All input and output streams, flowrates and compositions were measured except for the total flue gas flow rate. The flow rate of the flue gas was determined by closing the overall mass balance. The composition of the flue gas, shown in Table 6, is an average value of continuous on-line measurements.

The chemical analysis of different solid streams are shown in Table 7 to 10. The size analysis of these streams are given in the Appendix D. The composition of the bed material suggests that the circulating solids reached the steady state.

The average temperature of the combustor at the bottom, middle and top was 841°, 867° and 865°C respectively. The lower temperature at the bottom of CFBC was due to the incoming cold feed and primary air. The average standpipe temperature was 863°C, which, as expected, is close to the combustor top temperature.

TABLE 4

Summary of Combustor Experimental Conditions and Results

Solids Feed Rate:	21.2 kg/hr	C1
Coal: Spent Shale:*	1:1.5	
Air Feed Rate:	99.4 kg/Hr	F1-1
Excess Air:	8.1%	
Ca/S ratio:	0.62	
% Sulphur Reduction:	21.9%	
Combustion Efficiency:	90.1%	
Average Temperatures:		
Combustor Top:	865°C	T-20
Combustor Middle:	867°C	T-21
Combustor Bottom:	841°C	T-22
Standpipe:	863°C	T-23-25

* Coal: Spent Shale ratio used for all tests

TABLE 5

Combustor and Heat Exchanger Material and

Heat Balances Fifty Hours Duration**

Streams	Total Mass Flow Rate		Unit Enthalpy		Total Stream Enthalpy
	Symbol	kg/hr	J/kg	Sensor*	J/hr
COMBUSTOR					
Coal	C1	8.5	30.2×10^6	A*	257.0×10^6
Air***	FI-1	99.4	$.024 \times 10^6$	A*	2.38×10^6
Spent Shale	P2	12.7	1.61×10^6	A*	20.45×10^6
TOTAL ENTERING		120.6	-		279.83×10^6
Flue Gas****	FI-4	107.9	1.09×10^6	TC-11	117.3×10^6
Bed Discharge	C2	6.22	1.28×10^6		7.97×10^6
Secondary Cyc Discharge	C3	5.21	3.67×10^6		19.12×10^6
Baghouse Discharge	C4	1.19	1.39×10^6		1.66×10^6
Cooling Section		-	-		54.7×10^6
TOTAL LEAVING		120.5	-		200.75×10^6
HEAT EXCHANGER					
Hot Flue Gas Entering	FI-4	107.9	915.2×10^3	TC-11	98.75×10^6
Cold Flue Gas Leaving	FI-4	107.9	579.1×10^3	TC-10	62.49×10^6
Hot Recycle Retort Gas Leaving	FI-3	57.3	570.1×10^3	TC-13	32.7×10^6
Cold Recycle Retort Gas Entering	FI-3	57.3	58.2×10^3	TC-12	3.34×10^6

* A = Ambient temperature

** Fifty hours starting at the eighth hour of operation see Table 18

*** Measurement explained page 73

**** " " page 62 and Appendix F

TABLE 6

Average Dry Flue Gas Composition*

Component	Concentration
CO_2	14.6%
O_2	3.2%
CO	481 ppm
SO_2	3634 ppm
NO_x	208 ppm

* Average of 43 sets of readings taken about once every hour

TABLE 7

Chemical Analysis of Bed Discharge (C_2)

Sample No*	%C	%S	%Ash	%SiO_2	%Al_2O_3	%Fe_2O_3	%CaO	%MgO
1	1.0	0.99	99.3	74.9	8.77	4.83	3.09	1.19
2	1.7	0.60	98.2	70.9	12.77	6.81	3.78	1.34
3**	1.8	0.65	98.2	74.3	14.04	7.55	4.21	1.54
4	2.0	1.34	98.8	68.0	15.19	7.07	4.70	1.86
5	1.8	0.66	98.7	71.1	15.15	6.75	4.02	1.77
6	1.8	0.60	99.0	69.4	14.59	7.12	3.40	1.89
7	4.1	0.81	96.3	66.1	18.03	9.16	4.84	1.91
8	3.6	0.86	95.8	66.0	17.40	8.33	4.87	2.09

* 8 samples taken at 6 hour intervals

** Heating value for Sample No. 3 is 164 Btu/lb (3.8×10^5 J/kg)

TABLE 8

Chemical Analysis of Column Discharge (C_5)

Sample No*	%C	%S	%Ash	%SiO_2	%Al_2O_3	%Fe_2O_3	%CaO	%MgO
1	7.9	0.78	90.6	66.3	16.34	12.34	4.80	1.61
2	7.8	0.76	91.2	60.6	15.55	12.04	4.92	1.61
3**	5.8	0.73	93.6	62.4	13.38	14.36	4.97	1.43
4	6.1	0.86	93.3	64.7	14.21	11.75	4.88	1.75
5	8.3	0.77	90.9	60.3	13.24	11.41	4.69	1.61
6	6.5	0.73	92.6	68.0	12.87	11.74	4.29	1.49
7	6.5	0.79	92.6	61.0	12.13	12.26	4.27	1.38
8	8.4	0.74	91.8	61.7	13.38	11.87	3.76	1.38

* 8 samples taken at 6 hour intervals

** Heating value for Sample No. 3 is 891 Btu/lb (2.07×10^6 J/kg)

TABLE 9

Chemical Analysis of Secondary

Cyclone Discharge (C_3)

Sample No*	%C	%S	%Ash	%SiO_2	%Al_2O_3	%Fe_2O_3	%CaO	%MgO
1	11.3	1.00	86.6	60.9	12.04	12.18	4.49	1.46
2	11.4	0.91	87.5	61.0	11.11	13.76	5.04	1.51
3**	8.7	0.94	89.8	54.9	10.37	12.26	4.88	1.44
4	7.8	0.93	91.0	59.4	14.08	14.42	4.83	1.67
5	11.0	0.90	84.9	55.2	14.49	14.88	4.71	1.67
6	10.5	0.84	88.1	62.4	14.15	17.28	4.69	1.62
7	7.7	0.82	91.1	55.7	13.47	17.00	4.52	1.49
8	7.5	0.75	91.5	60.5	13.50	13.50	4.53	1.64

** Heating value for Sample No. 3 is 1,190 Btu/lb (2.77×10^6 J/kg)

TABLE 10

Chemical Analysis of Baghouse Solids (C_4)

Sample No*	%C	%S	%Ash	%SiO_2	%Al_2O_3	%Fe_2O_3	%CaO	%MgO
1	4.9	1.44	93.7	47.2	16.32	20.29	6.29	2.16
2	4.1	0.89	95.0	47.4	16.10	18.86	7.74	2.27
3**	3.7	1.17	95.1	46.7	16.40	18.65	6.93	2.29
4	3.3	0.97	95.4	51.8	16.78	17.80	6.34	2.55
5	3.8	0.75	95.3	47.9	16.36	17.02	7.05	2.60
6	2.9	0.84	96.3	45.9	16.89	17.49	11.67	2.45
7	3.1	0.95	95.7	49.9	16.42	17.90	6.02	2.35
8	2.8	0.43	96.2	50.2	16.00	17.15	5.67	2.14

* 8 samples taken at 6 hour intervals

** Heating Value for Sample No. 3 is 213 Btu/lb(4.95×10^5 J/kg)

7.2 The Retorting System

The entire continuous 57 hour run can be divided into three periods. During the initial period, which was 13 hours duration, there was some disruption of spent shale discharge which delayed reaching steady state operation. In the second period, steady state conditions were maintained in the entire system, for 20 hours. In the third period, the heating coil wrapped around the pipe leading from the heat exchanger to the retort inlet burnt out. This period is designated as the "Lower Heat Input" period. It was 22 hours in duration.

The summary of the experimental conditions and results is shown in Table 11. The residence time is calculated from the volumetric flow rate of shale and the volume of the retort. As indicated earlier the middle recycle was introduced to the retort at two levels, Table 11 shows the temperature range of the recycle gas during the run. Previous experience showed that a small amount of air at the bottom of the retort led to higher oil recovery. However, a high air input tended to decrease the oil yield which has been reflected in the lower oil recovery during the third period.

During the steady state period oil yield was high, the average recovery is 97.4% of the Fischer assay.* Material and heat balances for each period are shown in Table 12, 13 and 14. The material balance is quite satisfactory for all the periods, with the output slightly higher than the input for all three cases. Errors in retort exist gas measurment are the most likely explanation for this discrepency. The heat loss from the system which varied from 5.2 to 14.5%, is relatively low for a pilot scale unit indicating a well insulated reactor system.

* Yield is given in percentage of the Fischer assay (16.9 Imp gal/ton).

TABLE 11

Summary of Retort Experimental Conditions and Results

		Initial Period	Steady State	Lower Heat Input
Shale Feed Rate kg/hr	P1	18.0	18.0	18.0
Average Bed height, m		2.36	2.44	2.34
Shale Residence Time, hr		7.4	7.7	7.4
Total Recycle Gas, kg/100kg of Shale	FI-3	314.4	318.3	314.4
Recycle Gas to the Middle, %		76.1	78.2	79.2
Recycle Gas to the Bottom, %		23.9	21.8	20.8
Temperature of Middle Recycle Before entering (Upper)	T33/34	495-522	539-564	487-501
The Retort, ^{o}C (Lower)	T35/36	527-543	550-566	486-506
Air Feed to the Bottom kg/100kg of shale:		8.3	6.7	13.9
Oil Recovery, % of Fisher assay		76.0	97.4	75.2

TABLE 12

Material and Heat Balances Retorting System

Initial Period Thirteen Hours Duration

Retort	Total Mass Flow Rate Symbol	kg/hr	Unit Ethalpy J/kg	Sensor	Total Stream Ethalpy J/hr
Oil Shale	P1	18.0	5.575×10^6	A*	100.3×10^6
Recycle Retort Gas	FI-3	56.6	4.871×10^6		275.7×10^6
Air**		1.5	0.024×10^6		0.036×10^6
TOTAL ENTERING		76.1	-		376.04×10^6
Net Retort Gas**	FI-2	1.8	4.489×10^6		8.08×10^6
Retort Exit Gas		59.9	4.489×10^6	TC-9	268.9×10^6
Kerogen	FI-6	0.99	44.5×10^6	TC-9	44.1×10^6
Retort Cyclone Solids	P3	0.15	4.09×10^6	TC-9	0.61×10^6
Spent Shale	P2	14.0	1.75×10^6	TC-8	24.5×10^6
Water		0.8	3.96×10^5		0.32×10^6
TOTAL LEAVING		77.64			346.51×10^6
(IN-OUT)		- 1.54			29.53×10^6
% unaccounted		- 2.0			7.9

* Ambient temperature

** Small amount of air was metered with a rotameter and introduced into the retort with the bottom recycle gas

(page 62)

TABLE 13

Material and Heat Balances Retorting System

Steady State Twenty Hours Duration

Retort	Total Mass Flow Rate Symbol	kg/hr	Unit Ethalpy J/kg	Sensor	Total Stream Ethalpy J/hr
Oil Shale	P1	18.0	5.575×10^6	A*	100.3×10^6
Recycle Retort Gas	FI-3	57.3	4.388×10^6		251.4×10^6
Air**		1.2	0.024×10^6		0.029×10^6
TOTAL ENTERING		76.5	-		351.73×10^6
Net Retort Gas**	FI-2	1.8	3.996×10^6		7.19×10^6
Retort Exit Gas		59.9	3.996×10^6	TC-9	239.4×10^6
Kerogen	FI-6	1.34	44.7×10^6	TC-9	59.9×10^6
Retort Cyclone Solid	P3	0.15	4.12×10^6	TC-9	0.62×10^6
Spent Shale	P2	14.8	1.76×10^6	TC-8	26.05×10^6
Water		0.79	4.52×10^6		0.36×10^6
TOTAL LEAVING		78.78			333.52×10^6
(IN-OUT)		- 2.28			18.21×10^6
% unaccounted		- 2.9			5.2

* Ambient temperature

** Small amount of air was metered with a rotameter and introduced into the retort with the bottom recycle gas

(page 62)

TABLE 14

Material and Heat Balances Retorting System

Lower Heat Input Twenty Two Hours Duration

Retort	Total Mass Flow Rate Symbol	kg/hr	Unit Ethalpy J/kg	Sensor	Total Stream Ethalpy J/hr
Oil Shale	P1	18.0	5.575×10^6	A*	100.3×10^6
Recycle Retort Gas	FI-3	56.6	2.549×10^6		144.3×10^6
Air**		2.5	0.024×10^6		0.06×10^6
TOTAL ENTERING		77.1	-		244.66×10^6
Net Retort Gas**	FI-2	1.8	2.232×10^6		4.02×10^6
Retort Exit Gas		59.7	2.232×10^6	TC-9	133.3×10^6
Kerogen	FI-6	0.97	44.5×10^6	TC-9	43.2×10^6
Retort Cyclone Solids	P3	0.15	4.10×10^6	TC-9	0.62×10^6
Spent Shale	P2	16.3	1.69×10^6	TC-8	27.55×10^6
Water		0.79	4.52×10^5		0.36×10^6
TOTAL LEAVING		79.71			209.1×10^6
(IN-OUT)		- 2.61			35.56×10^6
% unaccounted		- 3.4			14.5

* Ambient temperature

** Small amount of air was metered with a rotameter and introduced into the retort with the bottom recycle gas

(page 62)

The distillation characteristics of the shale oil is shown in Figure 8. This fraction represents 87% of the total oil yield. The knock-out pot oil represents remaining 13% and its boiling characterisits is shown in Figure 9. All the distillation results are given in Appendix E (Table 25).

The elemental analysis and specific gravity of the shale oil and the product gas is shown in Table 15 and 16. The concentration of CO2 and CO in the retort gas was monitored continously and the hydrocarbons and H_2 were determined by gas chromotograph. Water has been determined by a taking slip stream and condensing the water. Hydrogen sulfide or oxygen could not be detected in the product gas.

The chemical anlaysis of the spent shale is shown in Table 17. This spent shale was fed to the combustor after mixing with Minto coal. The size analysis of the spent shale coal ash mixture is given in Appendix D (Tables 21-23). Size analysis of the spent shale from the retort is given in Table 24.

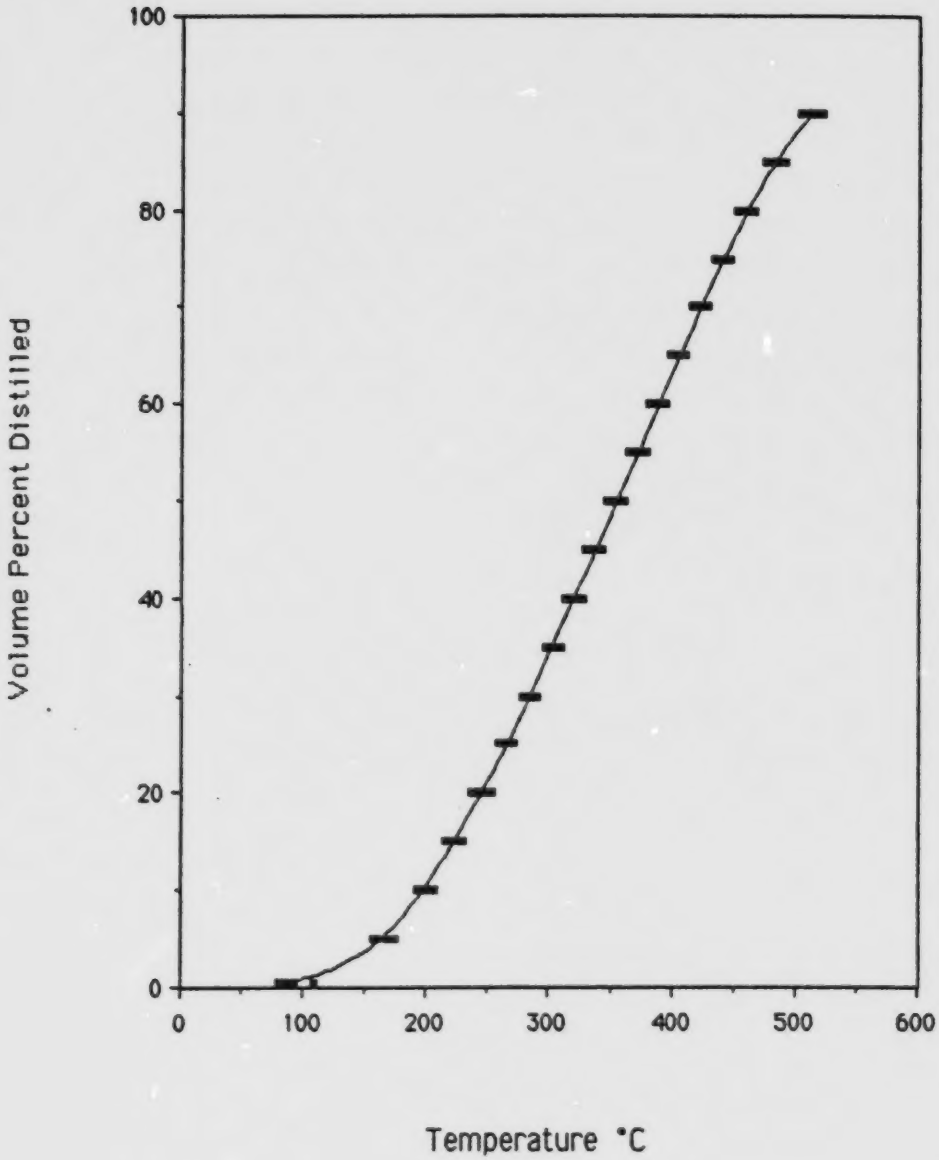

FIGURE 8: Simulated Distillation Analysis for the Condenser and ESP samples

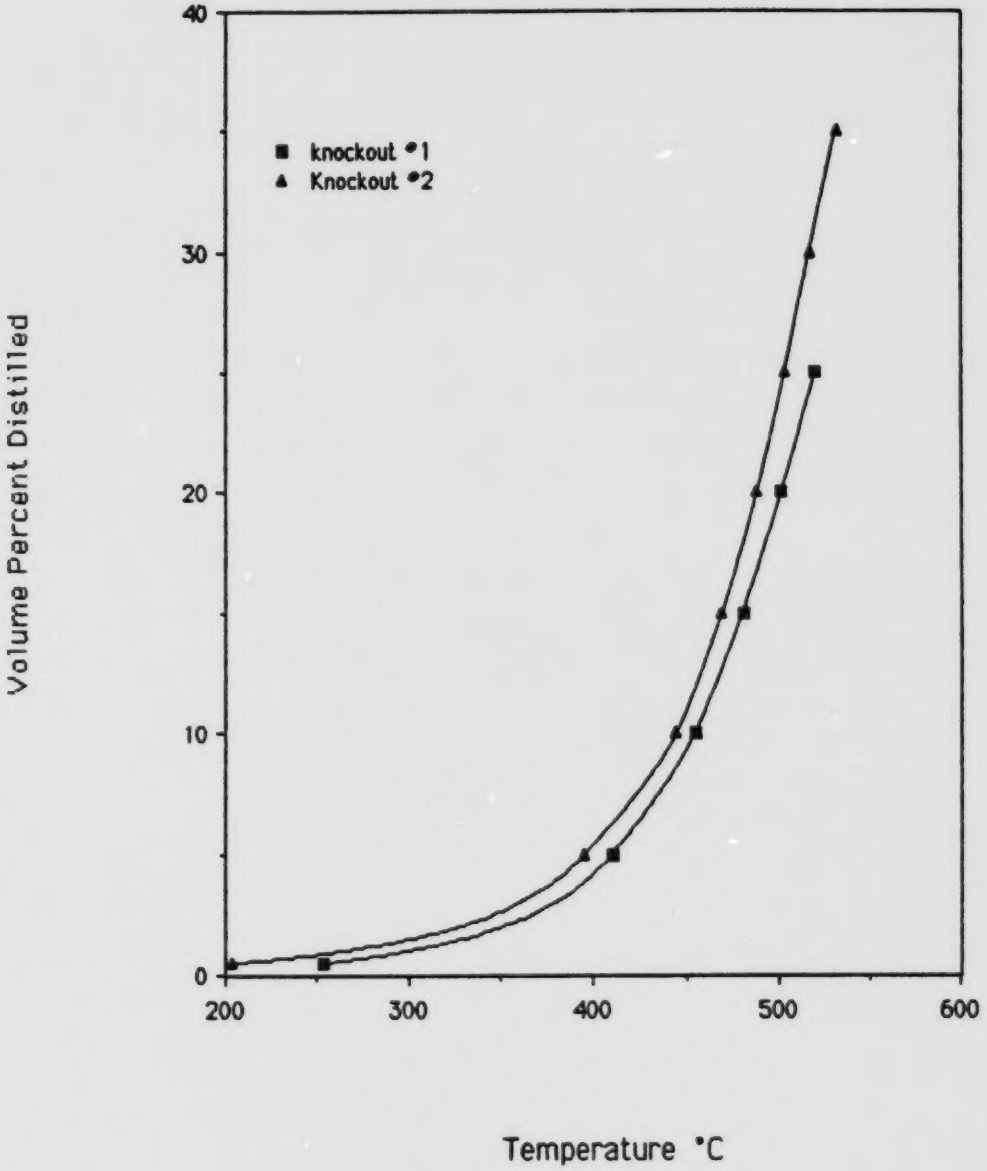

FIGURE 9: Simulated Distillation Analysis for the Knockout pot samples

TABLE 15

Chemical Analysis of Shale Oil

Sample No.	Gravity API	Elemental Analysis, wt % C	H	N	S	O	Ash	Sample Source
1	30.4	84.66	11.81	0.85	0.37	2.16	0.152	ESP, COND
2	32.3	85.93	11.71	0.85	0.40	1.01	-	ESP, COND
3	32.8	82.90	11.39	0.89	0.36	4.38	0.082	ESP, COND
4	32.1	84.85	11.63	0.81	0.39	2.07	0.054	ESP, COND
5	33.0	87.44	11.95	0.90	0.35	-	0.031	ESP, COND
6	33.6	85.16	11.94	0.59	0.36	1.94	0.006	ESP, COND
7	31.7	85.07	11.68	0.79	0.38	2.07	0.0010	ESP, COND
8	31.5	84.18	11.59	0.81	0.35	3.06	0.0009	ESP, COND
9	-5.1	67.63	8.79	0.91	0.41	1.99	20.27	Knock-Out
10	-15.8	62.38	7.59	0.91	0.42	6.10	22.60	Knock-Out

TABLE 16

Average Retort Gas Composition

	Initial Period		Steady State		Lower Heat In-Put	
	Wet	Dry	Wet	Dry	Wet	Dry
CO_2	5.57	5.7	5.49	5.6	10.07	10.3
CO	6.88	7.0	2.65	2.7	4.75	4.9
H_2	6.36	6.5	6.47	6.6	2.98	3.1
CH_4	5.40	5.5	5.30	5.4	2.52	2.6
C_2H_6	1.57	1.6	1.47	1.5	0.72	0.7
H_2O	1.92	-	1.91	-	2.06	-
N_2	72.3	73.7	76.7	78.2	76.9	78.5
Heating Value (Btu/SCF)		134	-	117	-	68
AV Mol. Wt.	26.43	26.60	26.18	26.56	28.34	28.56

Analyses used for the averages are given in Table 20.

TABLE 17

Chemical Analysis of Spent Shales

Sample No*	%C	%S	%Ash	%SiO_2	%Al_2O_3	%Fe_2O_3	%CaO	%MgO
1	5.8	0.43	89.8	57.3	14.95	14.13	4.63	1.77
2	5.8	0.28	90.5	67.1	12.58	13.96	4.36	1.76
3**	5.8	0.38	89.9	63.7	12.21	8.33	4.84	1.86
4	5.8	0.31	90.1	67.1	18.78	7.82	4.33	2.05
5	5.1	0.25	90.7	63.8	16.97	7.50	4.69	1.86
6	6.1	0.37	89.7	60.9	16.55	7.01	4.46	1.94
7	6.4	0.30	88.5	64.4	16.37	7.41	5.54	2.04
8	6.1	0.33	89.4	63.7	15.74	7.17	5.25	2.01
Average Value	5.86	0.33	89.8				4.76	
Knockout Solid	8.9	0.37	82.1	63.5	17.23	7.42	2.25	1.46

* The 8 samples are taken at six hour intervals

** Heating Value for Sample No. 3 is 693 Btu/lb(1.61×10^6)

8.0 DISCUSSIONS

8.1 Performance of the CFBC System

The combustor performed well during the entire run with no disruptions. The Ca/S ratio shown in Table 4 is very low resulting in low sulphur reduction. A typical Ca/S ratio for 90% sulphur capture is over 2:1. NO_x emission shown in Table 4 is typical of CFBC. The combustion efficiency of 90.1% is relatively low, however, high combustion efficiences could be achieved upon optimization of the operating conditions. In other tests combustion efficiencies of 95.3% were obtained (2). Scale-up will certainly improve the combustion efficiency.

Component balances such as ash balance, CaO balance, sulphur balance and carbon balances can be closed within 94% reflecting good analytical techniques and experimental measurements. The heat balances shown in Table 5 shows that the average calculated heat loss is 28.2%, which is large but not entirely unexpected due to the large surface area to volume ratio characteristics of all small reactors. This is especially true for CFBC systems which have a large standpipe containing inert solids.

8.2 Performance of the Retort

The temperature profile for the three periods are shown in Figures 10 to 12. The shapes of the temperature profiles are similar to previous retorting trials (1). Maximum temperature in the retort for best oil recovery did not exceed 570°C in any part of the retort. However, the best recoveries in the previous program was achieved at a maximum temperature of approximately 620°C. This temperature might be achieved with a heat exchanger of higher capacity. The temperature of the spent shale leaving the retort was low, which means that the sensible heat loss by the hot spent shale is not significant. The temperature of the spent shale

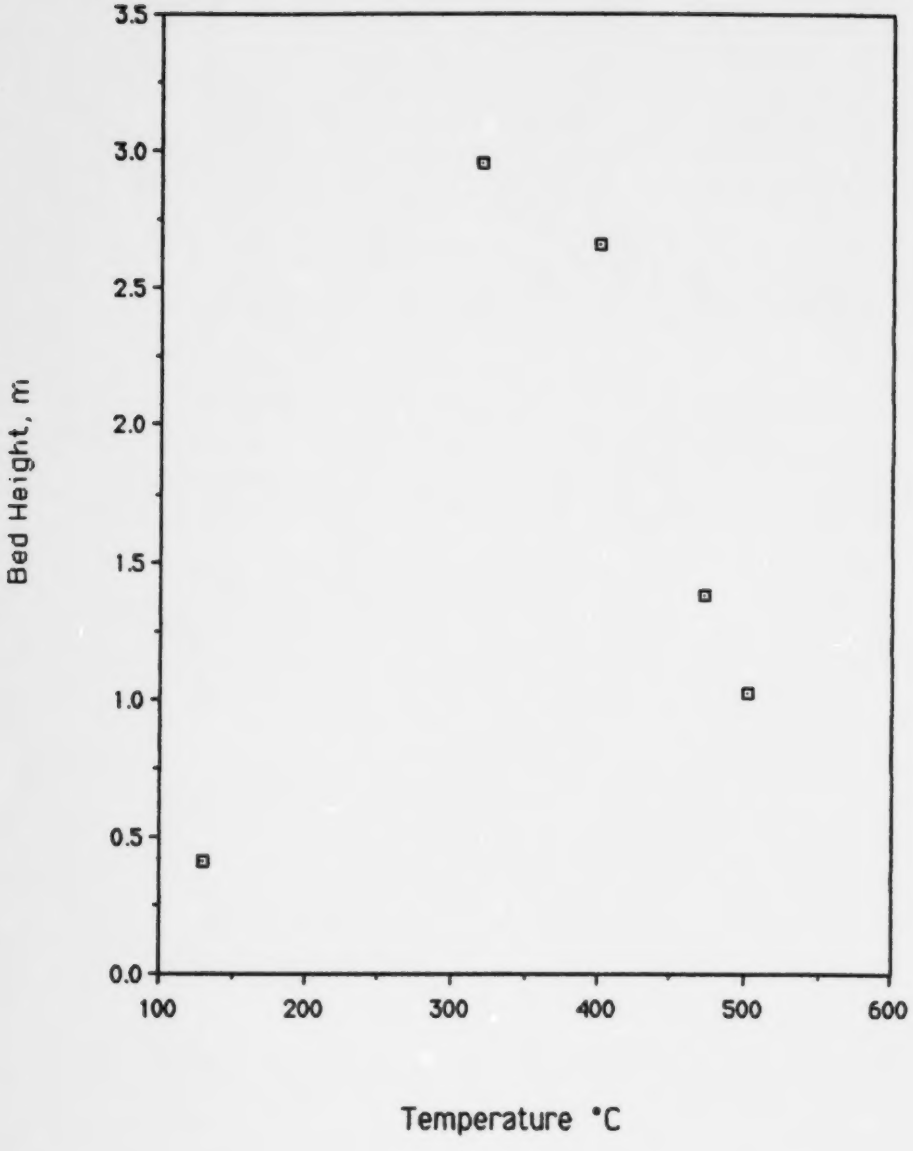

FIGURE 10: Temperature Profile in the Retort for the Initial Period

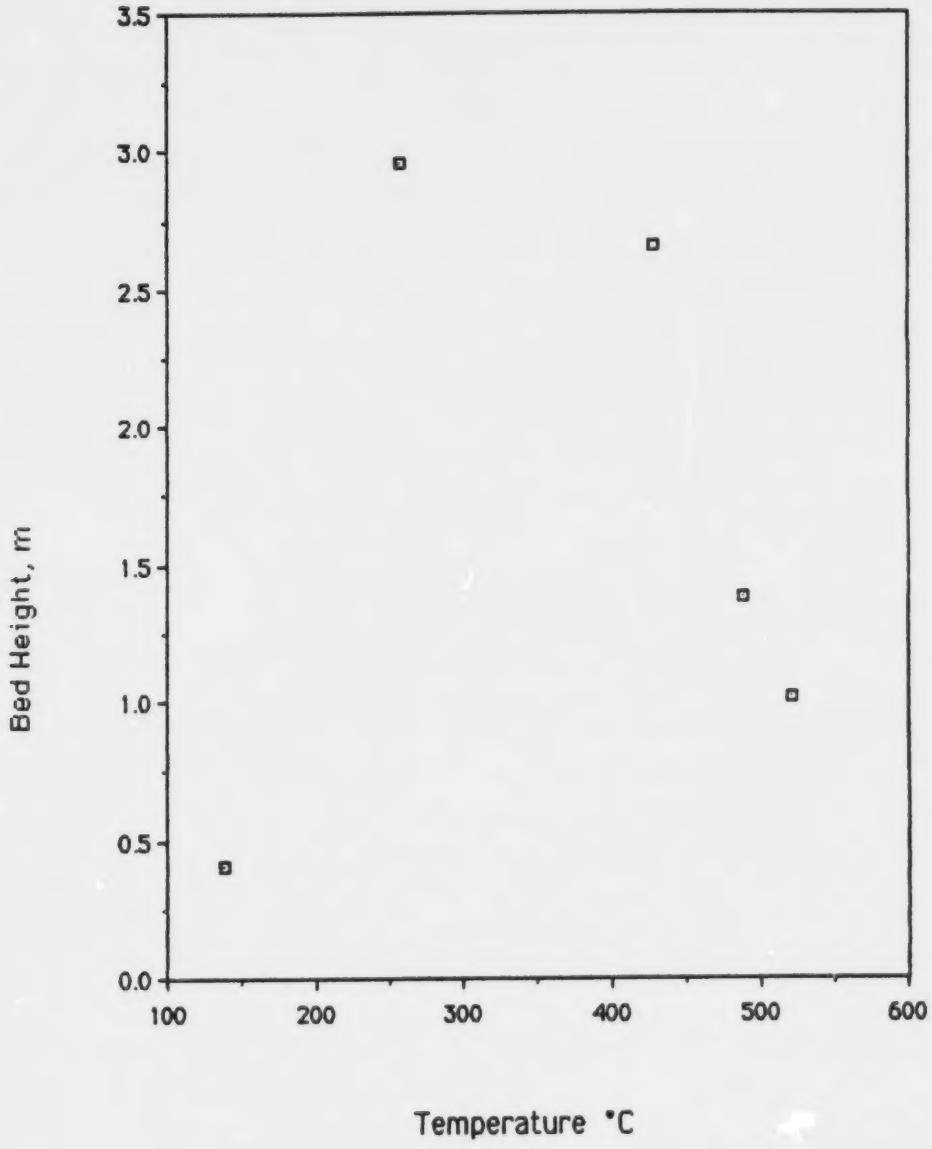

FIGURE 11: Temperature Profile in the Retort for the Steady State Period

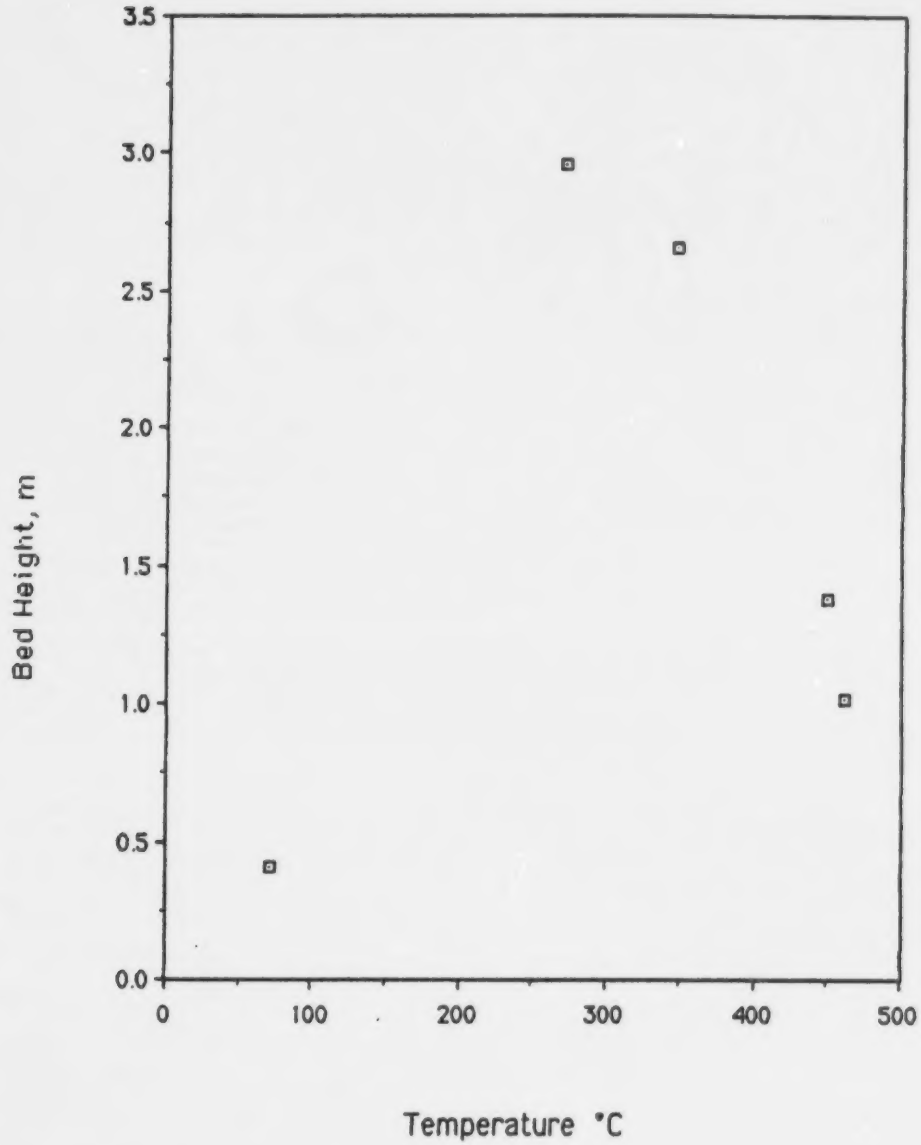

FIGURE 12: Temperature Profile in the Retort for the Lower Heat Input

leaving the retort was controlled by adjusting cold recycle gas at the bottom of the retort.

The heat input has a considerable effect on the oil yield as demonstrated by the lower yield in the third period. Yields dropped from 97.4% to 75.1% corresponding to a reduction in retorting temperature from about 550°C to 490°C due to failure of the heating coal. Also, the introduction of air for the generation of heat did not increase the oil yield.

A comparison of nitrogen and sulphur content of the shale oil, as shown in Table 14 with other shale oils including those obtained during the previous retorting program using N.B. oil shale shows that, these elements are consistantly lower than all previous results (1). However, oxygen determined by difference is significantly higher. The oil is lighter as reflected in the API gravity. The heavier nature of the knock-out pot oil is due to the presence of solid in the oil, as indicated by the higher ash content.

The calorific value of the gas ranged from 68 to 134 Btu/Scf. The lower calorific value during the third period was due to the increased air injection in the retort which was responsible for higher nitrogen content in the gas.

The calorific value of the spent shale was 693 Btu|lb, which translates to approximately 29% of the calorific value of the fresh shale.

In summary, the retort performed quite well during the steady state period with a good oil recovery. It may be possible to further improve oil recovery by optimizing different process parameters.

8.3 Performance of the Heat Exchanger

The outlet retort gas temperature from the heat

exchanger was not sufficient to adequately pyrolysis the oil shale. Therefore, the pipe leading from the heat exchanger to the retort inlet was wrapped with a heating coil. When this coil was working as in the steady state period, the oil recovery was high.

The heat exchanger performed according to the design specifications. However, it was designed for smaller amounts of recycle retort gas than were used in these runs. A higher rate of recycle would be needed to increase retorting temperatures for better oil recovery.

9.0 CONCLUSIONS

Steady state oil yield during retorting was 97.4% of Fisher assay. The heat input to the retort has strongly effected the oil yield.

Oil quality in terms of nitrogen and sulphur content is better than any other shale oil, including previously retorted N.B. shale oil. The oil is as light as a Kuwaiti crude (1). The oxygen content of the oil, determiend by difference, is significantly higher than previous results.

The calorific value of the product gas, during the steady state was 134 Btu/SCF, which compares favourably with our previous retorting results. The spent shale retained over 20% of the calorific value of the fresh shale.

The combustor performed smoothly for the entire duration of the test. The combustion efficiency of the spent shale and Minto coal is consistent with other fuels tested in this system (2). Sulphur capture was low because of the lower Ca/S molar ratio. For better sulphur capture, the ratio of spent shale to coal must be increased. NO_x emission level of about 200 ppm is quite satisfactory.

The heat exchanger performed according to the design specifications. However, the temperature of the middle recycle had to be boosted further for better oil recovery. Improved oil yield could be obtained by redesigning the heat exchanger.

10.0 RECOMMENDATIONS

The effects of different operating variables on the integrated combustion/retorting process have not been investigated in this program. All the operating variables were set based on our previous experience. It is felt that the oil yield, combustion efficiencies, sulphur capture etc. can be improved by optimizing the different operating parameters. It is recommended that such a program be initiated.

The heat exchanger should be modified or redesigned to adequately preheat the middle recycle gas.

11.0 REFERENCES

1. P.F. Salib, S.K. Barua, E. Furimsky, "Retorting of Oil Shale from New Brunswick, Canada", The Canadian Journal of Chemical Engineering, Vol. 64, No. 6, 1986.

2. P.F. Salib, S.K. Barua, K.R. Robson, R.K. Kissell, "Circulating Fluidized Bed Combustion of different Ranking Coals with Limestone Sorbents", presented at the Nineth International Conference on Fluidized Bed Combustion, Boston, Mass., May 3-7, 1987.

DEPARTMENT: APPLIED SCIENCES

PROJECT TITLE: INTEGRATED RETORTING COMBUSTION SYSTEM USING FIXED BED TECHNOLOGY

PROJECT NUMBER: PROPOSAL NUMBER

REPORT AS/87/26

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APPENDIX A

Work Statementt

AN INTEGRATED RETORTING-COMBUSTION SYSTEM USING FIXED BED TECHNOLOGY

BACKGROUND

A proposal was received from Monenco/RPC in response to a "Request for Proposal," RFP, entitled, "Process Evaluation of an Integrated Retorting Co-Combustion System. In order to evaluate the experimental capability of the contractor more precisely, it has been suggested that the contractor perform a single experiment of a comprehensive nature. It is intended that the contractor's performance during the course of this work will be evaluated and used to rank his capability for performing other future work involving integrated pyrolysis/combustion. The possibility of continuing this work in a second phase, is an option which will be considered later.

The conditions of the experiment have been chosen in an attempt to demonstrate the contractors capability of fullfilling the requirements of the above RFP. The experiment is described in the attached work statement.

OBJECTIVE

The objective of the work is to perform a steady state experiment in which the heat for pyrolyzing oil shale is obtained indirectly from coal combustion. Specifically

1. An integrated pyrolysis-combustion system is used. That is both the pyrolysis reactor and the combustion reactor are to operate simultaneously for the duration of the experiment.
2. New Brunswick Oil Shale is used for pyrolysis
3. Minto coal is used for combustion
4. The steady state portion of the experiment has a duration of at least 48 hours
5. The fluids from the retort are separated from the solids and cooled to produce a retort gas stream and a retort liquid stream
6. A portion of the retort gas stream is heated by the hot combustor flue gas stream and then recycled to the pyrolysis reactor, where it is used to provide the heat for pyrolysis.

The project manager should be responsible for all aspects of the contract. The principal investigator should be personally responsible for the performance of the experimental work.

STATEMENT OF WORK

Task 1 Materials - Procurement and Analysis

The contractor is to obtain the amounts of New Brunswick Oil Shale and Minto coal required to perform the work. The following analyses shall be performed.

Coal - proximate
ultimate
heating value
mineral matter analyses
SiO_2 -m%
Al_2O_3 - m %

Fe_2O_3 - m %
MgO - m %
CaO - M %
particle size analysis

Oil Shale - proximate
ultimate
Fischer assay
mineral matter analyses
SiO_2 - m %
Al_2O_3 - m %
Fe_2O_3 -m %
MgO - m %
CaO - m %
particle size analysis

Task 2 Suitability of Existing Equipment

The existing experimental equipment in the contractor's laboratory shall be compared

(a) with the block diagram shown in Figure 1 and
(b) with the conceptual flow sheet shown in Figure 2

The contractor shall prepare a flow sheet for the equipment to be used to perform the experiment. It shall either

(a) conform to the block diagram and conceptual flow sheet above

or

(b) be a modification of the above, such that the objectives of the work are accomplished. Any modification must be approved by the scientific authority in writing, prior to the commencement of the work.

Task 3 Compatibility of Process Variables with Existing Equipment

The desired process variables, listed in Table 1 shall be compared with the capability of the existing experimental equipment. The contractor shall recommend specific values for these process variables, which will produce optimum liquid yields.

Preliminary material and heat balances shall be made based on the contractor's past experience. Examples of these calculations are, (1) the coal feed rate to the combustor shall be chosen to provide sufficient heat to pyrolyze the Oil Shale using the indirect method (heat exchange with the retort gas). (2) The recycle retort gas flow rate shall be chosen to give efficient particle removal in the cyclone. After considering the above calculations the scientific authority and the principal investigator shall mutually agree on the processing conditions to be used in Task 5.

Prior to beginning the experimental work in Task 5, the contractor must obtain written approval from the scientific authority, for the processing conditions to be used.

Task 4 Equipment Modification

Based on the results from Tasks 2 and 3, a list of required equipment modifications shall be prepared,

and the cost of performing these modifications shall be estimated. The calculated operating conditions are sometimes different than those actually required. As a result the equipment should be capable of operating at a variety of operating conditions where possible. This flexibility should be considered in finalizing the list of required equipment modifications. Written approval from the scientific authority shall be obtained prior to making any equipment modifications. In the event that the cost of the required modifications are excessive, the contract may be cancelled at this point.

Task 5 Equipment Operation

The experimental unit shall be started-up in the configuration shown in the approved flow sheet (Task 2). The unit shall be brought to the steady-state processing conditions identified in writing (Task 3). The unit shall be operated at these steady-state conditions for a minimum duration of 48 hours. During the steady-state period,
-the measurements shown in Table 2 shall be made every hour
-the measurements shown in Table 3 shall be made every 6 hours.
-the samples listed in Table 4 shall be taken every 6 hours.
The analytical determinations listed in Table 5 shall be made on the respective samples.

Operational points to be implemented during the experiment include the following:
-to prevent condensation of hydrocarbons in the cyclone, the cyclone should be kept at the same temperature as the retort.
-the combustor should be operated with 5 % excess air
-the retort solids bed should contain only oil shale (no sand)
-the combustor solids bed should contain only coal and spent shale (no sand)
-an experimental measurement of solids residence time in the retort must be made
-an experimental measurement of solids residence time in the combustor must be made
-liquids, gases and solids are to be produced as separate streams and their yields measured.

Task 6 Material and Heat Balance Calculations

Typical measured values from the experimental work in Task 5 shall be used to complete material and energy balances for the approved process flowsheet in accordance with the format of Tables 6 and 7. The amounts of spent solids which originated from the shale and from the coal can be determined from material balances on iron and magnesium, assuming the only change in the ash as it flows through the retort and the pyrolyzer is the conversion of calcium to the sulphate form. The estimated heat loss from the experimental unit will be particularly important for scale-up purposes.

Task 7 Final Report

The final report shall consist of:
-a figure showing the approved flowsheet
-a table listing the approved processing conditions
-Table 2 showing all the measured experimental values taken at 1 hour intervals

-Table 3 showing all the measured experimental values taken at 6 hour intervals
-results of all the analytical determinations as listed in Table 5
-Tables 6 and 7 showing the results of all the material and heat balance calculations
-specifications of all the experimental equipment (line sizes, vessel dimensions, exchanger areas, etc)
-itemized procedures used for
start-up
operating at steady-state
shut-down
emergencies
-a narrative description of events which occurred during the start-up, steady-state operation and shut-down of the experimental equipment.

One draft copy of the final report is to be provided to the scientific authority.

Five (5) copies of the approved final report are to be provided to the scientific authority.

An oral presentation of the final report shall be delivered in Ottawa.

CONTRACT PLAN AND REPORT FORM

-duration of each task and
-cost of each task
is to be specified by the contractor.

The Contractor's performance during the course of the work will be evaluated by comparing it with the Statement of Work in the Contract.

APPENDIX B

Guide to Experimental Data

TABLE 1
Operating Conditions

Fig	Pg	Table	Pg	Primary Operating Conditions
4,6	19,21	11-14,19a	30-33,75	oil shale feed rate, P1
4,6	19,21	19a	75	retort temperature, TC-1 (top), TC-8 (btm)
4	19	18a	64	coal feed rate, C1
4,5	19,20	18a	64	combustor temperature, TC-20 (top), TC-22 (btm)
				Secondary Operating Parameters
		11	30	solids residence time in the retort gas residence time in the retort
6	21	11,12,14	30,31,33	retort gas recycle rate, FI-3 solids residence time in the combustor gas residence time in the combustor
5	20	18c	70	air flow to combustor, FI-1

TABLE 2
Experimental Measurements at
1 Hour Intervals

Fig	Pg	Table	Pg	
4,6	19,21	11-14,19a	30-33,75	oil shale feedrate, P1
6	21	12-14	31-33	recycle retort gas flowrate, FI-3 start-up nitrogen for pyrolyzer, FI-5
4,6	19,21	12-14	31-33	net retort gas flowrate, FI-2
4,6	19,21	19a	75	retort top temperature, TC-1
4,6	19,21	19a	75	retort bottom temperature, TC-8
4,6	19,21	19a	75	retort cyclone temperature, TC-9 temperature of cooled retort fluids, TI-6
4,6	19,21	19b	78	temperature of cold retort recycle gas, TC-31
6	21	19c	81	temperature of hot retort recycle gas, TC-33-36
4-6	19-21	18b	67	temperature of cold flue gas, TC-10
4-6	19-21	18b	67	combustor cyclone temperature, TC-11
4,5	19,20	18a	64	combustor top temperature, TC-20
4,5	19,20	18b	67	standpipe bottom temperature, TC-25
4	19	18a	64	coal feed rate, C1
6	21	5	25	flue gas flow rate, FI-4
5	20	18c	70	combustor air flowrate, FI-1

TABLE 3

Experimental Measurements at 6 Hour Intervals

Spent Shale Flowrate From Retort, P2

(from mass balances, Fe, Mg)

Fig	Pg	Table	Pg	
		12,13	31,32	retort cyclone solids flow rate, P3
6	21	12,14	31,33	kerogen flowrate, FI-6
				coal ash solids flowrate, C2 (from mass balances, Fe, Mg)
7	22	18a	64	combustor cyclone solids flowrate, C3

TABLE 4

Process Samples at 6 Hour Intervals

Fig	Pg	Table	Pg	
4,6	19,21	12-14,19a	31-33,75	spent shale, P2
		12,14	31,33	retort cyclone solids, P3
6	21	12,14	31,33	retort gas
6	21	11,12,14	30,31,33	kerogen or retort liquid
		18a	64	spent solids
7	22	18a	64	combustor cyclone solids, C3
	22	18c	70	flue gas

TABLE 5

Analytical Determinations Made on Process Samples

Fig	Pg	Table	Pg	
		7,8,9,10	27,28	1. Spent Shale, 2. Retort Cyclone Solids 3. Spent Solids and 4. Combustor Cyclone Solids
				total elemental C and S - m%
				ash - m%
		17	39	mineral matter analyses
				SiO_2 - m%
				Al_2O_3 - m%
				Fe_3O_3 - m%
				MgO - m%
				CaO - m%
		21-24	87-90	particle size analysis
		19c	81	Kerogen or retort liquid
8,9	35,36	24	91	curve of boiling point versus % distilled gravity
		15	37	elemental C, H, N, S, and O
				m % ash
				Retort gas
		16	38	H_2, H_2S, N_2, O_2, CO_2, NO_x, CO, CH_4,
		20	84	C_2, C_3, and C_4
				Flue gas
		18b	67	N_2, O_2, CO, CO_2, NO_x, and SO_2
		6	26	

APPENDIX C

Raw Data

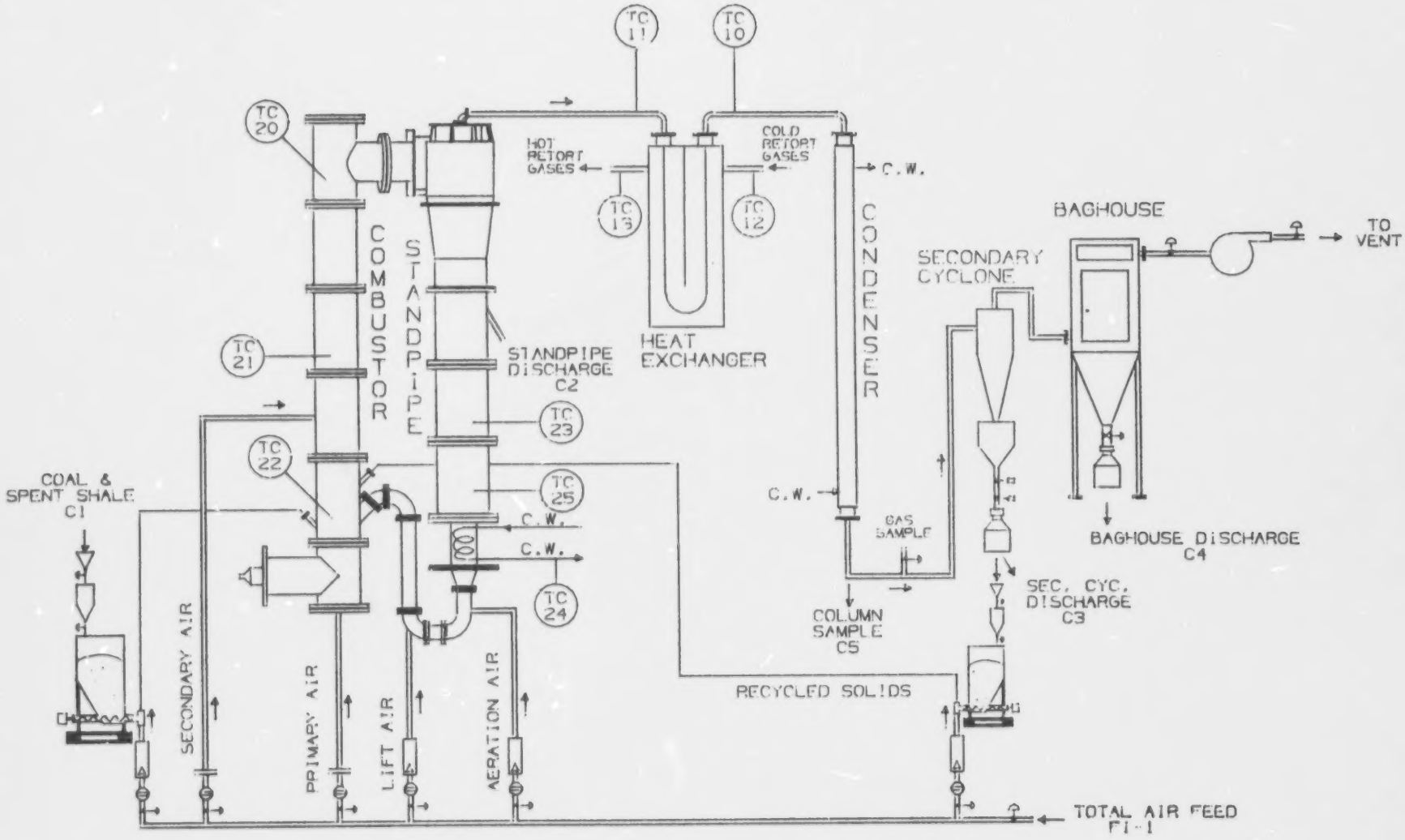


FIGURE 5 : SCHEMATIC OF RPC CIRCULATING FLUIDIZED BED COMBUSTOR SHOWING STREAM SYMBOLS AND INSTRUMENTS FOR TEMPERATURE AND FLOW MEASUREMENTS

Experimental Measurements

Experimental measurements taken at one hour intervals are given in Tables 18a-c and 19a-c for the combustor and retort, respectively. Abbreviations used in the tables are explained in Appendix H.

In accordance with Table 2 of the work statement measurements to be taken at 1 hour intervals are as follows:

Item	Measurement Taken	
	Yes	No
- oil shale feed rate (P1)	X	
- recycle retort gas flow rate (FI-3)	X	
- start up nitrogen (FI-5	X[1]	
- net retort gas flow rate (FI-2)		X[2]
- retort top temperature (TC1)	X	
- retort bottom temperature (TC8)	X	
- retort cyclone temperature (TC9)	X	
- temperature of cooled retort fluids	X[3]	
- temperature of cold retort recycle gas (TC12)	X	
- temperature of hot retort recycle gas (TC13)	X	
- temperature of cold flue gas (TC10)	X	
- combustor cyclone temperature (TC11)	X	
- combustor top temperature (TC20)	X	
- standpipe bottom temperature (TC25)	X	
- coal feed rate (C-1)	X[4]	
- flue gas flow rate (FI-4)	X[5]	
- combustor air flow rate (FI-1)	X	

1 No nitrogen was used during start-up.

2 Net retort gas flow rate (FI-2) was measured on an intermittent basis by diverting the flow from the stack to a rotameter. Continuous measurement was not possible due to fouling of the rotameter with the residual oil in the gas stream. This stream can also be determined by subtracting the sum of the middle recycle and the bottom recycle from the total gas flow, see Table 19B.

3 Thermocouples 31, 32 and 30 record recycle gases temperature, immediately after the recycle blower (TC#31) and at the middle and bottom recycle orifice plates. Liquid products temperature was not monitored on a continuous basis, however, it was close to ambient temperature.

4 The combustor was fed with a coal/spent shale mixture at a ratio of 1:1.5 i.e. coal feed rate is 40% of the mixture feed rate.

5 Flue gas flow rate (Table 5) was calculated from an elemental mass balance on the combustor using nitrogen as a tie component. Direct measurements can be difficult because of the relatively high dust loading in the flue gases. Details of the calculations are given in Appendix F.

Experimental measurements taken at six hour intervals as per Table 3 of the work statement are as follows:

Item	Measurement Taken Yes	No
- retort cyclone solids flow rate (P3)		X[1]
- kerogen flow rate (F-6) **Note:** This was measured every hour as marked above.	X	
- coal ash solids flow rate (C2/C3/C4) **Note:** These are the combustor solids discharged from the standpipe/bed, secondary cyclone and the baghouse. They were measured every hour.	X	
- combustor cyclone solids (C2) **Note:** Measured every hour as itemized above.	X	

1. Retort cyclone (knockout pot) products are a tar like material mixed with very fine dust. For this reason knockout pot solids flow rate was not measured on a regular basis. An estimation of the solids content of the knockout pot product was made on a composite sample of the knockout pot products, extracting the liquid products using a solvent. Chemical analysis of the solids are shown in Table 17 while simulated distillation analysis of the knockout pot liquid products is given in Figure 9.

TABLE 18A

Combustor Operating Conditions and Results

time (hr)	Coal Spent Shale C1 feed rate (kg/hr)	recycled solids (kg/hr)	C2 stnp dis (kg/hr)	secondary C3 cyc dis (kg/hr)	baghouse C4 discharge (kg/hr)	temp bot TC # 22 (°C)	temp mid TC #21 (°C)	temp top TC #20 (°C)
1			0.3	3.9	0.7	828	849	848
2		4.0		5.2		841	864	864
3	9.4	5.3		6.1		842	868	868
4	8.6	6.2	1.3	6.0	1.8	826	847	846
5	9.2	5.9		9.1		849	845	845
6	22.9	3.8		7.9		846	871	869
7	8.8		1.3	6.7	2.3	703	730	732
8	**24.0**	**1.7**	**0.2**	**4.1**		**818**	**843**	**844**
9	22.6	2.2	0.2	6.0		840	867	868
10	22.0	5.8	0.2	5.9	3.5	830	855	855
11	21.0	7.6	4.0	6.1		827	854	853
12	21.4	5.7	4.0	6.6	2.9	828	856	856
13	22.0	6.1	8.6	10.4	1.1	831	859	859
14	21.4	5.2	9.2	6.8	1.5	841	868	868
15	**23.2**	**0.3**	**4.6**	**2.0**	**1.1**	**827**	**852**	**854**
16	22.4	6.3	12.9	8.0	2.0	823	858	858
17	22.8	6.7	8.1	5.0	1.4	833	866	866
18	22.2	6.2	6.7	5.6	1.3	838	867	869
19	19.6	6.1	5.6	4.9	1.2	836	862	861
20	20.4	5.3	6.4	5.2	0.4	842	869	868

(See Figure 5)

	Coal Spent Shale			secondary	baghouse	temp bot	temp mid	temp top
	C1		C2	C3	C4			
time (hr)	feed rate (kg/hr)	recycled solids (kg/hr)	stnp dis (kg/hr)	cyc dis (kg/hr)	discharge (kg/hr)	TC # 22 (^{o}C)	TC #21 (^{o}C)	TC #20 (^{o}C)
21	19.0	5.3	5.7	5.5		845	874	874
22	16.4	4.7	0.3	4.2		833	858	857
23	19.8	0.5	6.4	3.6	3.1	843	872	869
24	20.4	7.8	8.4	5.3		851	875	873
25	21.4	6.3	4.0	5.3	2.2	846	870	867
26	17.8	5.0	7.8	4.9		852	876	874
27	18.4	5.5	9.3	3.9		851	878	875
28	20.0	5.0	6.8	4.2		840	860	858
29	21.6	5.4	16.8	4.7	2.3	841	863	859
30	20.8	6.1	4.1	4.4				
31	20.2	6.0	6.5	5.3	2.3	838	860	857
32	24.2	6.3	2.8	5.1		850	873	872
33	22.4	6.4	7.9	4.6	2.3	837	865	865
34	22.5	6.3	1.4	5.6		836	862	863
35	**23.3**	**6.6**	**6.8**	**6.7**	**3.4**	**844**	**869**	**867**
36	22.2	6.6	3.8	6.4		832	859	858
37	21.4	5.4	6.5	5.6	2.4	826	852	852
38	21.4	5.8	5.9	5.7			856	855
39	22.0	5.1	6.4	5.2	3.5	836	863	863
40	21.6	5.8	7.4	5.2		849	872	870

time (hr)	Coal Spent Shale C1 feed rate (kg/hr)	recycled solids (kg/hr)	C2 stnp dis (kg/hr)	secondary C3 cyc dis (kg/hr)	baghouse C4 discharge (kg/hr)	temp bot TC # 22 (°C)	temp mid TC #21 (°C)	temp top TC #20 (°C)
41	21.2	8.0	5.0	5.4		845	867	865
42	20.8	6.9	6.2	5.3	4.3	840	864	862
43	21.2	6.2	6.9	4.8	1.4	843	865	863
44	21.6	5.6	7.5	4.9		843	864	862
45	23.6	5.4	6.7	4.5	2.2	842	867	866
46	17.2	4.8	8.1	4.7		847	871	869
47	21.5	4.8	4.6	4.6		844	866	861
48	20.5	0.0	9.6	3.5	4.2	852	879	875
49	20.2	6.8	5.9	5.5	1.3	846	869	867
50	21.0	5.0	6.3	4.8		851	876	875
51	20.4	4.4	5.2	4.5		840	864	863
52	20.6	4.5	7.6	4.5	4.5	844	867	866
53	21.2	4.6	6.6	5.3		851	873	870
54	22.2	4.7	6.7	5.2		853	876	874
55	21.4	5.0	7.1	5.1	3.9	856	874	874
56	22.0	4.5	9.2	4.7		867	892	890
57	21.2	4.0	5.7	4.8	3.2	868	892	890

TABLE 18B

Standpipe and Heat Exchanger Operating Conditions and Results

Time (hr)	stnp mid TC #23 (°C)	stnp bot TC #25 (°C)	ht exch in TC #11 (°C)	ht exch out TC #10 (°C)	cool out TC #24 (°C)	Flue Gas Analysis CO2 (%)	O2 (%)	CO (ppm)	SO2 (ppm)	NOx (ppm)
1	826	831	680	410						
2	850	856	707	423	24					
3	851	856	703	426	32	13.2	5.0		4200	
4	834	839	681	408	31	13.0	5.2	710	4100	
5	838	837	697	416	40	12.4	5.4		3800	
6	854	859	714	421	40					
7	743	713	623	412	34					
8	**835**	**830**	**702**	**419**	**84**	**14.2**	**3.3**	**600**	**4700**	**170**
9	860	719	428	857						
10	853	849	727	437	52					
11	853	850	717	433	46	14.4	3.0	530	3700	190
12	855	851	718	430	44	14.8	3.1	610	3750	182
13	856	851	722	432	43	14.9	3.1	650	4100	179
14	859	863	727	458	31	15.2	2.7	700	4200	185
15	**856**	**851**	**719**	**493**	**41**	**15.0**	**3.0**	**700**	**4300**	**190**
16	855	850	733	481	44	15.0	3.2	620	3500	140
17	862	845	730	459	44	15.8	2.4	950	4300	120
18	864	856	729	448	46	15.8	2.5	600	3900	145
19	861	854	729	443	43	15.8	2.5	670	4100	Note 2
20	865	859	731	439	44	15.6	2.4	520	4500	

(See Figure 5)

Time (hr)	stnp mid TC #23 (°C)	stnp bot TC #25 (°C)	ht exch in TC #11 (°C)	ht exch out TC #10 (°C)	cool out TC #24 (°C)	Flue Gas Analysis CO2 (%)	O2 (%)	CO (ppm)	SO2 (ppm)	NOx (ppm)
21	870	862	732	437	44	15.4	2.6	590	4200	Note 2
22	860	858	720	432	43	14.8	3.5	500	4100	
23	869	858	723	433	44	14.6	3.2	520	3800	
24	870	861	713	435	44	14.4	3.2	450	4300	
25	868		755	437	40	14.4	3.9	400	3400	
26	875	866	754	435	39	15.0	3.3	400	4100	
27	874	866	Note 1	436	41	14.8	3.2	350	3900	
28	860	854		436	41	13.0	5.0	410	2400	
29	858	846		433	41	12.8	5.0	450	2500	
30					40					
31	848	835		428	40	14.2	4.4	480	2800	195
32	864	852		430	38	15.0	2.8	460	3700	240
33	863	857		434	58	14.7	2.8	450	3100	
34	857	847		439		14.6	3.0	500	3700	154
35	864	856		450	53					
36	858	Note 1		446	52	14.5	3.2	380	3300	210
37	853			448	49					
38	859			449	48	14.8	3.0	450	3600	190
39	836			450	51	15.2	2.4	400	4100	200
40	867			450	42	15.0	2.6	460	3900	230

Note 1: Thermocouple malfunction

Note 2: Temporary problem with NOx analyzer

Time (hr)	stnp mid TC #23 (°C)	stnp bot TC #25 (°C)	ht exch in TC #11 (°C)	ht exch out TC #10 (°C)	cool out TC #24 (°C)	Flue Gas Analysis CO2 (%)	O2 (%)	CO (ppm)	SO2 (ppm)	NOx (ppm)
41	865			452	45	14.6	3.0	480	3100	220
42	863			449	44	14.8	3.2	470	3200	200
43	864			449	42	14.4	3.4	450	3300	210
44	863			450	44	14.2	3.2	450	3400	215
45	866			452	42	14.6	3.4	440	3200	250
46	867			453	40	14.2	3.1	370	3600	240
47	862			457	41	14.0	3.5	370	3200	230
48	870			457	40	14.0	3.7	450	3200	248
49	868			461	42	14.4	3.5	440	3200	230
50	870			455	40	14.0	4.0	350	3200	255
51	862			455	41					
52	864			482	42	14.2	3.5	430	3300	240
53	867			461	42	14.2	3.7	270	3300	260
54	868			462	41	14.3	3.3	310	3700	240
55	872			465	39	14.5	3.2	310	3700	250
56	881			466	37	14.5	3.3	300	3700	240
57	883			465	38					

TABLE 18C
Combustor Air Feed Rate

Time (hr)	Primary G. Press (Psig)	Primary Diff Press inches/water	Aeration % Flow %	Aeration G. Press (Psig)	Lift % Flow %	Lift G. Press (Psig)	Feeder % Flow %	Feeder G. Press (Psig)	Sec. Cyclone Rec. % Flow %	Sec. Cyclone Rec. G. Press (Psig)
1	4.2	45	35	2-5	35	2-5	60	6.1	25	2.2
2	4.0	45	35	2-5	35	2-5	60	6.0	25	2.2
3	4.0	45	35	2-5	35	2-5	60	6.0	25	2.2
4	3.8	45	35	2-5	35	2-5	60	6.0	25	2.0
5	4.0	45	35	2-5	35	2-5	60	6.0	25	2.0
6	4.2	45	35	2-5	35		60	6.0	25	
7	4.2	45	35		35		60		25	
8	**4.2**	**45**	**35**	**2-5**	**35**	**2-4**	**60**	**6.3**	**25**	**2-5**
9	4.1	45	35		35		60		25	
10	4.2	45	35		35		60		25	
11	4.2	45	35		35		60		25	
12	4.3	45	35		35		60		25	
13	4.2	45	35		35		60		25	
14	4.2	45	35		35		60		25	
15	**4.3**	**45**	**35**		**35**		**60**		**25**	
16	4.2	45	35	2-5	35	2-4	61	6.2	25	2.6
17	4.2	45	35	2-5	35	2-4	60	6.2	25	2.6
18	4.3	45	35	2-5	35	2-4	60	6.3	25	2.6
19	4.2	45	35	2-5	35	2-4	60	6.2	25	2.6
20	4.1	45	35	2-5	35	2-4	61	6.1	25	2.6

Time (hr)	Primary G. Press (Psig)	Primary Diff Press inches/water	Aeration % Flow %	Aeration G. Press (Psig)	Lift % Flow %	Lift G. Press (Psig)	Feeder % Flow %	Feeder G. Press (Psig)	Sec. Cyclone Rec. % Flow %	Sec. Cyclone Rec. G Press (Psig)
21	4.0	45	35	2-5	35	2-4	61	6.2	25	2.4
22	4.0	45	35	2-5	35	2-4	61	6.2	25	2.2
23		45	35	2-5	35	2-5	61	6.0	25	2.1
24	4.0	45	35	3-5	35	3-5	60	6.2	25	2.1
25	4.0	45	35	2-5	35	3-5	60	6.0	25	2.0
26	4.0	45	35	2-5	35	1-4	60	6.0	25	2.0
27	3.9	45	35	2-5	35	1-4	60	6.0	25	2.0
28	3.9	45	35	2-5	35	1-4	60	6.2	25	2.1
29	3.8	45	40	2-5	40	1-4	60	6.0	25	2.2
30	4.2	44	50		40		60		25	
31	3.8	45	40		35		60		25	
32	3.8	45	35	2-4	35	1-3	60	6.0	25	
33		45	35		45		60		25	
34		45	25		45		60		25	
35	4.1	45	25		45		60		25	
36		45	25		45		60			
37	4.2	45	25		45		60			
38	4.1	44	45		25		60			
39	4.1	44	45		25		60			
40	4.3	45	45	2-4	25	2-4	60			

Time (hr)	Primary G. Press (Psig)	Primary Diff Press inches/water	Aeration % Flow %	Aeration G. Press (Psig)	Lift % Flow %	Lift G. Press (Psig)	Feeder % Flow %	Feeder G. Press (Psig)	Sec. Cyclone Rec. % Flow %	Sec. Cyclone Rec. G. Press (Psig)
41		45	45		25		60		25	
42		45	45		25		60		25	
43		45	45		25		60		25	
44		45	45		25		60		25	
45		45	45		25		60		25	
46	4.6	45	48		25		60		25	
47	4.4	45	45		25		60		25	
48	4.4	45	45		25		60	6.2	25	2.4
49	4.2	45	45		25		60	6.0	25	2.2
50	4.4	45	45		25		60		25	
51	4.3	45	45		25		60	6.0	25	2.1
52	4.3	45	45		25		60		25	
53	4.3	45	45		25		60		25	
54	4.3	45	45		25		60		25	
55	4.2	45	45		25		60		25	
56	4.2	45	45		25		60		25	
57	4.3	45	45		25		60		25	

Total Air Feed Rate FI-1

Total air supply to the combustor consisted of 5 different streams as shown in Figure 5 and Table 18C. These are primary, aeration, lift, feeder and secondary cyclone recycle. Measurement was hourly. The primary air stream was measured using a square-edged orifice plate of an I.D. of 0.600" with flange taps mounted on a 1.5" Sch.40 steel pipe. Aeration, lift, feeder and secondary cyclone recycle streams were measured using identical rotameters calibrated as follows:

100% = 8.33 CFM, Sp. Gr. 1.0 when metered at 10 psig and 70°F.

As shown in Table 18C measurements were hourly. There is little variation and individual measurements were not recalculated. An average was taken before making the conversion by the method shown in Appendix G.

It is important to mention that the wide fluctuations in aeration and lift air pressures (1 to 5 psig) are normal for J-valve applications.

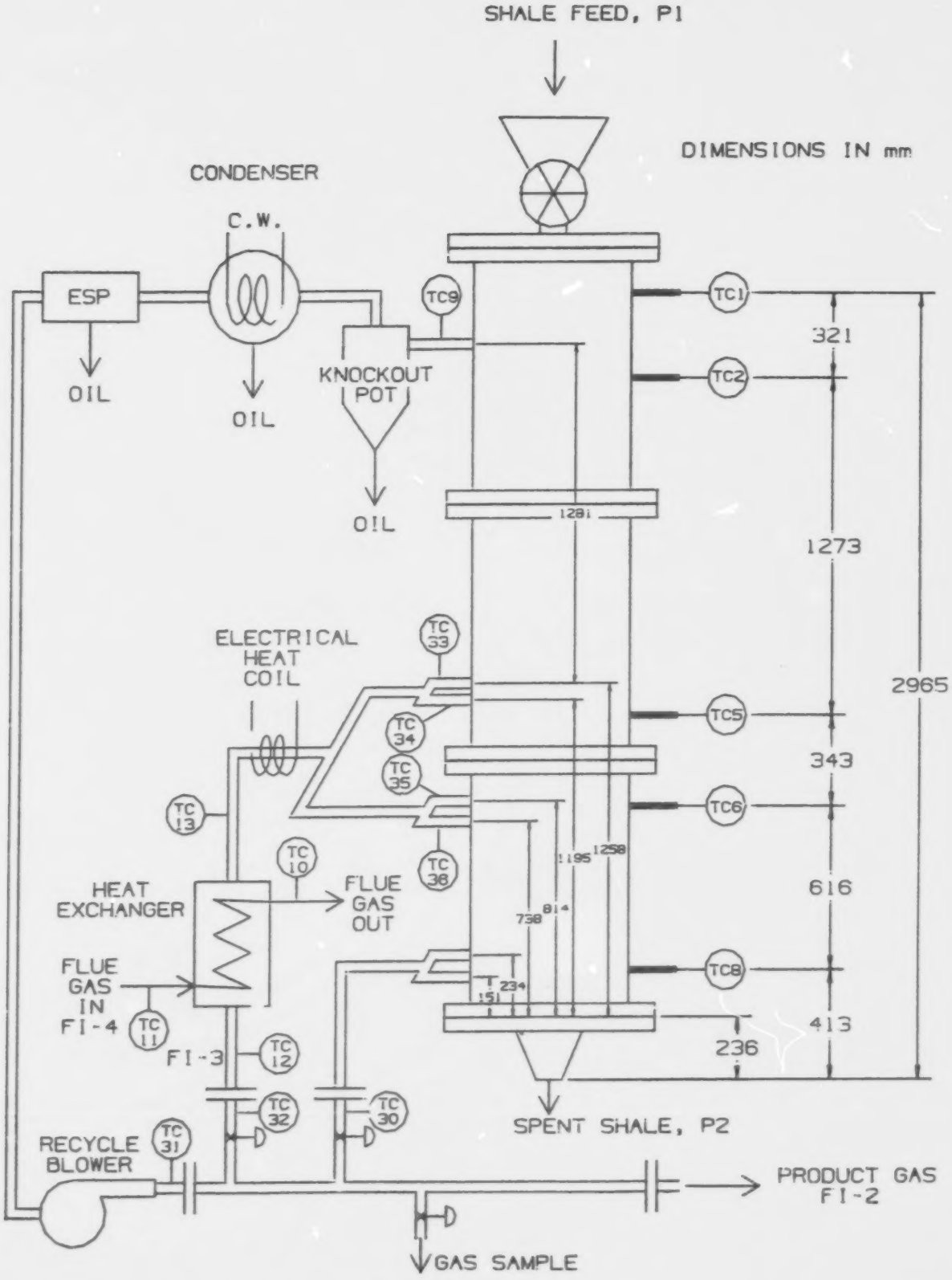


FIG 6 : SCHEMATIC OF RPC FIXED BED RETORT SHOWING STREAM SYMBOLES AND INSTRUMENTATION FOR TEMPERATURE AND FLOW MEASUREMENTS

TABLE 19A
Retort Operating Conditions and Results

time (hr)	shale feed rate, P1 (kg/hr)	bed height (in)	spent shale dis, P2 (kg/hr)	Retort temperature profile TC # 8 (°C)	TC #6 (°C)	TC #5 (°C)	TC #2 (°C)	TC #1 (°C)	gas exit TC #9 (°C)	Heat Exchanger In TC #T12 (°C)	Heat Exchanger Out TC #13 (°C)
1	19	93	15.1	150	493	445	373	183	119	52	438
2	19	97	15.4	161	500	451	380	158	191	53	453
3	19	93	14.5	149	504	466	389	248	207		455
4	18	99	14.3	148	494	472	394	368	219	54	433
5	18	93	14.7	139	493	469	398	377	195	55	440
6	18	94	14.8	155	499	469	395	377	216	54	456
7	18	94	14.3	169	502	474	399	380	203	53	428
8	18	94	12.4	169	501	477	403	153	242	54	443
9	18	92	13.8	168	508	473	404	368	263	54	456
10	18	94	12.5	96	510	479	406	372	234	54	465
11	18	88	13.1	64	508	485	416	387	223	53	459
12	18	88	12.9	55	499	483	414	397	263	53	457
13	18	91	12.1	68	506	484	425	397	136	53	459
14*											
15											
16	18	93	14.0	182	524	435	366	262	263	49	497
17	18	97	13.2	103	535	491	401	212	199	49	472
18	18	97	14.7	109	528	504	414	188	172	49	463
19	17	96	15.8	137	525	497	422	180	149	49	459
20	17	95	15.7	135	526	498	424	187	190	50	459

* A temporary problem with discharge screw at the 14th hour.
No data was recorded for the 14th or 15th hours.
(See Figure 6)

time (hr)	shale feed rate, P1 (kg/hr)	bed height (in)	spent shale dis, P2 (kg/hr)	Retort temperature profile TC # 8 (°C)	TC #6* (°C)	TC #5* (°C)	TC #2 (°C)	TC #1 (°C)	gas exit TC #9 (°C)	Heat Exchanger In TC #T12 (°C)	Out TC #13 (°C)
21	18	97	15.9	132	520	496	423	196	222	50	456
22	18	96	15.6	142	520	495	428	265	275	50	450
23	18	95	16.1	182	522	492	422	241	267	50	454
24	18	93	18.4	191	525	488	417	247	270	51	458
25	18	91	14.7	176	523	490	413	297	286	52	459
26	18	97	8.3	102	526	483	457	322	262	52	458
27	18	99	16.3	140	529	497	449	228	266	53	460
28	18	97	14.2	131	526	497	439	140	242	53	458
29	18	96	16.5	121	522	496	432	316	295	52	451
30	18	97	15.3	131	518	498	436	223	262	53	449
31	18	99	15.6	128	516	493	441	315	294	53	446
32	18	99	14.7	128	517	493	442	136	218	53	450
33	18	93	15.1	136	521	494	444	395	233	53	454
34	18	93	8.9	100	523	458	460	433	218	53	459
35	**18**	**91**	**16.3**	**151**	**478**	**471**	**422**	**386**	**153**	**51**	**468**
36	18	91	12.5	126	451	454	416	389	248	53	467
37	18	100	13.5	84	460	444	409	387	232	53	470
38	17	103	16.9	75	459	442	407	384	251	52	468
39	17	104	17.6	83	456	431	403	381	221	52	469
40	16	104	17.9	82	458	439	396	373	154	51	468

* Note the drop in the retort temperature after the 35th hour due to the loss of electrical heat.

time (hr)	shale feed rate, P1 (kg/hr)	bed height (in)	spent shale dis, P2 (kg/hr)	Retort temperature profile TC # 8 (°C)	TC #6 (°C)	TC #5 (°C)	TC #2 (°C)	TC #1 (°C)	gas exit TC #9 (°C)	Heat Exchanger In TC #T12 (°C)	Out TC #13 (°C)
41	16	100	18.7	76	454	442	392	292	171	52	468
42	17	96	17.3	75	451	438	396	216	218	52	465
43	18	94	18.8	72	458	439	393	200	171	52	464
44	18	90	18.1	75	455	443	392	249	197	52	464
45	18	91	16	62	467	452	393	239	228	52	468
46	18	90	17.2	62	467	456	394	224	209	51	470
47	18	89	16.8	64	461	455	252	276	174	50	472
48	18	89	16.7	57	466	457	200	225	188	52	470
49	18	87	17.3	66	459	459	239	257	268	53	477
50	18	86	16.3	71	469	461	236	230	227	54	469
51	18	83	15.4	61	446	453	250	251	254	53	469
52	18	84	14.5	62	468	458	204	227	238	53	469
53	18	91	10.6	51	468	450	398	226	235	52	473
54	18	88	17.7	60	471	448	396	279	283	53	476
55	18	84	16.5	72	471	451			252	53	477
56	18	88	11.9	61	476	454	324	191	189	53	482
57	18	87	16.5	61	476	454	396	210	155	53	481

TABLE 19B
Retort Recycle Gases

Time (hr)	Total Gas Flow* Diff. Press (In. water)	G. Press (psig)	Temp TC# 31 (°C)	Middle Recycle** Diff. Press (In. water)	G. Press (psig)	Temp TC#32 (°C)	Bottom Recycle*** Diff. Press (In. water)	G. Press (psig)	Temp. TC#30 (°C)
1	22.4	4.7	94	10	3.6	63	45	10	45
2	22.5	4.6	93	10	3.4	64	6.5	0.9	47
3	22	4.4	94	10.5	3.2	66	6.0	0.8	44
4	23	5.0		11.5	3.8		6.5	1.2	
5									
6	22.5	4.8	96	10	3.7	65	6	1.0	47
7	21.5	4.7	102	9.0	3.7	6.6	6	1.5	47
8	23	5.2	96	11.2	4.0	66	6.0	1.2	46
9	23	46		11.0	3.4		6.0	0.6	
10	23	4.8	93	11.1	3.6	66	6.0	0.7	47
11	23	4.7		11.1	3.7		6.0	0.5	
12	23	4.7	86	11.0	3.5	6.3	6.0	0.5	48
13	23	5.3	93	11.0	4.0	6.5	6.0	1.0	46
14									
15									
16	24	4.8	85	13	3.5	59	5.5	0.9	40
17	23.5	4.4	77	12.5	3.2	57	5.3	0.8	41
18	23	4.7	78	12.5	3.4	57	5	0.8	41
19	23	4.4	79	12.0	3.2	57	5	0.8	41
20	23	4.6	79	12.0	33	58	5	0.8	42

* Square-edged orifice, 0.67" ID, Flange taps, 1.5" Sch. 40 pipe

** Square-edged orifice, 0.649" ID, Flange taps, 1" Sch. 40 pipe (See Figure 6)

*** Square-edged orified, 0.472" ID, Flange taps, 1" Sch. 40 pipe

Time (hr)	Total Gas Flow* Diff. Press (In. water)	G. Press (psig)	Temp TC# 31 (°C)	Middle Recycle** Diff. Press (In. water)	G. Press (psig)	Temp TC#32 (°C)	Bottom Recycle*** Diff. Press (In. water)	G. Press (psig)	Temp. TC#30 (°C)
21	23	4.7	82	12.5	3.4	60	5	0.9	43
22	23	4.2	81	12.5	3.2	58	5	0.6	42
23	22	4.4	85	12	3.2	60	5	0.8	42
24	23.5	4.7	83	12	3.6	60	5	1.0	43
25	23	4.7	84	12.5	3.5	61	5	1.0	43
26	22.5	4.6	85	12	3.4	61	5	1.0	43
27	22.5	4.6	87	11.5	2.9	63	5	0.5	45
28	22	4.4	87	11.5	3.2	62	5	1.0	45
29	22.5	4.5	86	11.5	3.4	62	5	1.1	45
30	23	4.6	86	11.5	3.4	62	5	1.1	45
31	23	4.6	86	11.5	3.4	62	5	1.0	46
32	23	4.6	87	12	3.9	63	5	1.0	96
33									
34	23		91	12		63	5		44
35	**23**	**4.6**	**86**	**12**	**3.3**	**64**	**5**	**1.1**	**43**
36	23	4.2	87	11.1	3.0	63	5	1.2	42
37	23	3.9	90	11.5	2.8	63	5	0.8	47
38	24	4.7	88	12.5	3.4	62	5.5	1.3	48
39	23.5	4.5	87	12.5	3.4	62	5.5	1.2	48
40	24	4.8	86	12.5	3.7	61	6	1.4	47

Time (hr)	Total Gas Flow* Diff. Press (In. water)	G. Press (psig)	Temp TC# 31 (°C)	Middle Recycle** Diff. Press (In. water)	G. Press (psig)	Temp TC#32 (°C)	Bottom Recycle*** Diff. Press (In. water)	G. Press (psig)	Temp. TC#30 (°C)
41	24	4.8	85	12	3.4	61	6	1.3	47
42	23.5	4.8	86	12	3.3	61	6	1.4	47
43	23.5	4.7	87	12	3.4	61	6	1.4	48
44	24	4.9	87	12.5	3.6	62	6	1.4	48
45	23.5	5.1	89	12.5	3.8	62	6.0	1.6	49
46	23.5	5.0	89	12.5	3.8	60	6	1.6	48
47	24	5.0	89	12	3.6	60	6	1.4	48
48	23.5	4.9	92	12	3.6	61	6	1.5	48
49	23.5	4.9	91	12	3.6	62	6	1.5	49
50	23	4.6	91	10	3.4	64	5	1.2	49
51	23	4.6	93	11	3.5	64	5	1.4	49
52	23	4.6	94	11.5	3.6	63	5	1.4	48
53	22.5	4.7		10.5	3.4		5	1.3	
54	23	4.7	95	11.2	3.5	64	5	1.3	48
55	23	49	95	11.5	3.7	63	5	1.3	48
56	23	4.8	96	10.5	3.6	63	5	1.4	49
57	23	4.6	95	12.0	3.4	64	5	1.2	48

TABLE 19C
Retort Recycle Gases and Oil Yield

time (hr)	TC #33 (°C)	TC #34 (°C)	TC #35 (°C)	TC #36 (°C)	cond oil (ml)	cond. H_2 (ml)	Esp oil (ml)	total** oil (ml)	% Fischer* assay
1	536	520	491	512	340	680	1035	1460	90.7
2	543	525	496	519	300	760	780	1165	72.4
3	546	529	500	525	300	900	990	1375	85.4
4	531	516	489	512	280	620	850	1215	79.7
5	531	517	496	513	260	860	870	1215	79.7
6	542	525	497	521	230	900	740	1055	69.2
7	543	523	490	519	220	920	740	1045	68.5
8	540	523	485	519	160	640	830	1075	70.5
9	552	534	493	530	220	800	780	1085	71.2
10	553	537	500	533	220	700	710	1015	66.6
11	550	538	504	532	280	720	870	1235	81.0
12	533	502	528	150	660	170	405		6.6
13	547	532	496	526	170	660	460	715	46.9
14									
15									
16	586	564	552	586	1460	790	125	1670	109.5
17	585	570	557	582	810	790	150	1045	68.5
18	576	560	547	574	950	640	510	1545	101.3
19	572	556	543	570	340	940	1225	1650	114.6
20	567	553	540	565	360	900	1225	1670	116.0

** Total oil is the sum of the condensor and electrostatic precipitator oil plus the knock-out pot heavy oil which has an average value of 85 ml/hr

* % Fischer assay: % total oil/Fischer yield (16.9 Imp gal/ton, Table 2)

time (hr)	TC #33* (°C)	TC #34* (°C)	TC #35* (°C)	TC #36* (°C)	cond oil (ml)	cond. H_2O (ml)	Esp oil (ml)	total oil (ml)	% Fischer assay
21	564	550	539	562	340	720	1120	1545	101.3
22	560	546	535	558	400	920	1155	1640	107.6
23	568	551	540	566	300	730	730	1115	73.1
24	570	554	543	568	380	830	1460	1925	126.3
25	568	553	541	566	380	720	1100	1565	102.6
26	569	552	543	565	360	850	830	1275	83.6
27	571	555	544	568	320	800	870	1275	83.6
28	570	553	543	569	300	750	830	1215	79.7
29	566	550	540	562	290	860	1100	1475	96.7
30	561	545	533	560	340	700	1050	1475	96.7
31	560	544	534	558	320	860	1000	1405	92.2
32	562	546	536	561	400	700	1040	1525	100.0
33	566	551	540	565	350	800	1020	1455	95.4
34	567	550	541	567	200	700	620	905	59.4
35	**510**	**496**	**493**	**508**	**1860**	**790**	**190**	**2135**	**140.0**
36	482	469	467	481	240	790	220	545	35.7
37	491	476	474	489	150	790	125	360	23.6
38	495	480	477	491	500	650	150	735	51.0
39	496	481	478	492	420	710	710	1215	84.4
40	495	480	478	491	320	760	1140	1545	114.0

* Note the drop in the recycle gas input temperature after the 35th hour due to the loss of electrical heat.

time (hr)	TC #33 (°C)	TC #34 (°C)	TC #35 (°C)	TC #36 (°C)	cond oil (ml)	cond. H_2O (ml)	Esp oil (ml)	total oil (ml)	% Fischer assay
41	495	481	479	492	300	1240	900	1285	94.8
42	493	479	476	490	260	700	900	1245	86.5
43	497	481	480	494	300	750	860	1245	81.7
44	497	481	480	493	320	700	860	1265	83.0
45	506	489	487	502	260	600	880	1225	80.3
46	509	492	490	503	290	800	730	1105	72.5
47	510	494	491	507	220	800	830	1135	74.4
48	509	492	491	505	300	820	960	1345	88.2
49	512	495	492	508	270	880	980	1335	87.6
50	512	496	493	509	250	800	850	1185	77.7
51	502	484	482	496	240	780	600	925	60.7
52	510	493	491	506	280	820	650	1015	66.6
53	516	497	493	509	200	700	495	780	51.2
54	521	503	500	515	200	860	595	880	57.7
55	522	504	497	516	200	760	865	1150	75.4
56	526	506	503	519	190	750	520	795	52.1
57	529	511	507	522	210	860	690	985	64.6

TABLE 20

Dry Retort Gas Analyses

Hours into test run	8	11	Av	16	23	Av	30	34	39	45	Av
CO_2	5.8	5.6	5.7		5.6	5.6	9.5	10.0	10.6	11.0	10.3
CO	7.0	7.0	7.0		2.6	2.6	5.8	5.9	5.5	2.2	4.8
H_2	6.5	6.5	6.5	6.7	6.5	6.6	4.2	3.3	2.4	2.3	3.1
CH_4	5.6	5.4	5.5	5.5	5.2	5.4	3.3	2.7	2.3	1.9	2.6
C_2H_6	1.7	1.6	1.6	1.6	1.4	1.5	1.0	0.8	0.6	0.5	0.7

Averages presented in Table 16

APPENDIX D

Size Analysis of Different Solid Samples

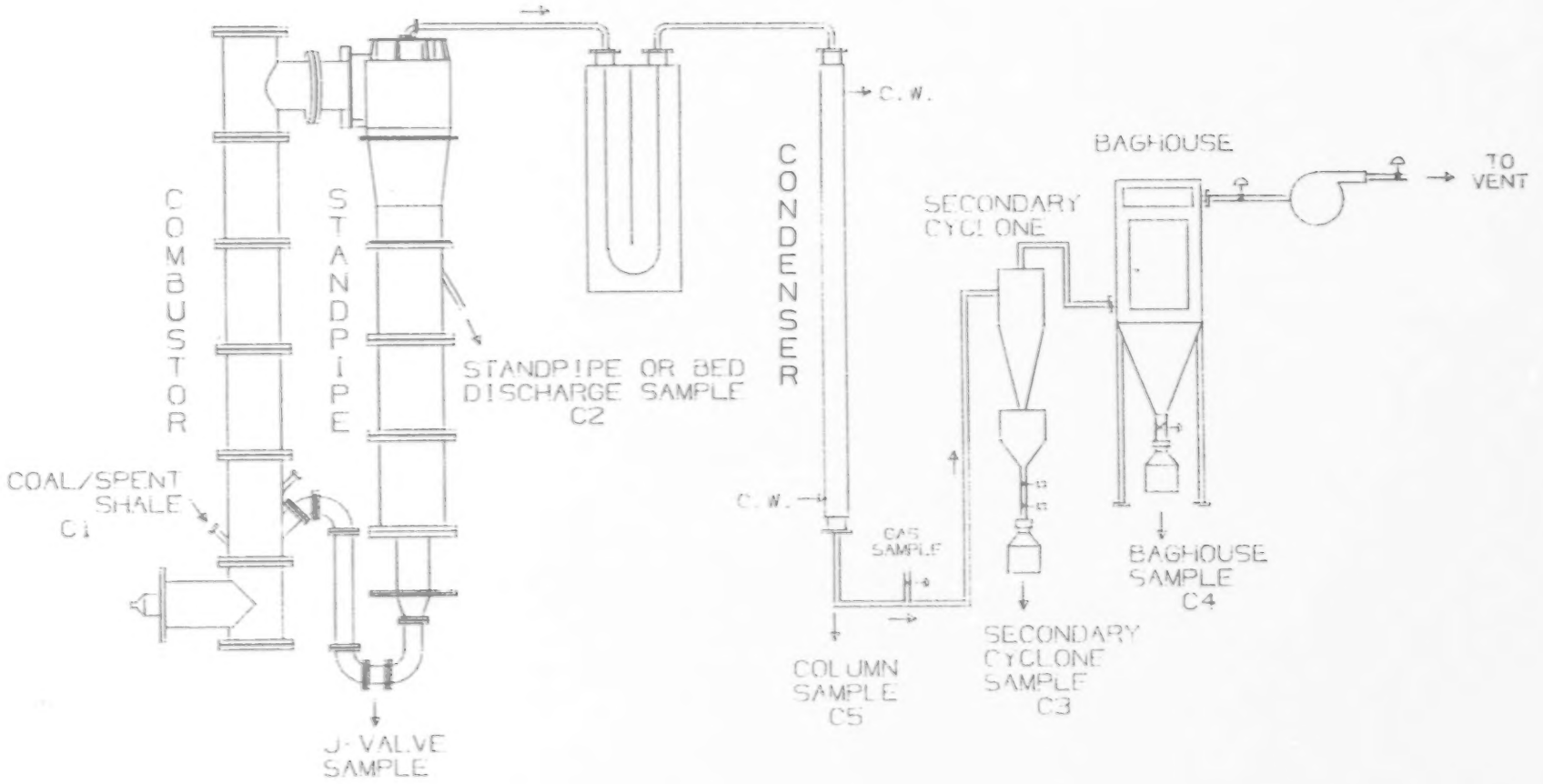


FIGURE 7 : SCHEMATIC OF RPC CIRCULATING FLUIDIZED BED COMBUSTOR SHOWING SOLID SAMPLING LOCATIONS

TABLE 21

Size Analysis of Solid

Samples from Standpipe

Sample No.	Weight, %				
	+1.4 (mm)	+0.71 (mm)	+0.355 (mm)	+0.1 (mm)	-0.1 (mm)
1	9.3	23.8	35.4	21.7	9.7
2	7.6	14.5	22.4	27.9	27.5
3	6.3	11.0	23.6	28.3	30.8
4	12.6	42.6	18.6	24.3	1.9
5	6.1	10.6	20.2	28.4	34.7
6	4.4	7.2	18.6	30.6	39.2
7	9.8	13.0	16.0	22.2	40.0
8	12.5	13.7	18.3	27.3	28.1

TABLE 22

Size Analysis of Solid

Samples from Column

Sample No.	Weight, %				
	+1.4 (mm)	+0.84 (mm)	+0.59 (mm)	+0.4 (mm)	-0.42 (mm)
1	8.3	9.5	11.9	30.2	40.1
2	7.4	7.8	9.5	26.6	48.7
3	8.7	7.3	10.8	33.2	40.1
4	5.9	7.6	9.1	29.3	47.9
5	6.8	7.8	10.4	30.8	44.2
6	4.4	4.5	6.6	30.8	53.7
7	3.2	3.5	6.1	32.3	54.8
8	3.7	4.4	8.7	31.6	51.6

TABLE 23

Size Analysis of Solid

Samples from Secondary Cyclone

Sample No.	Weight, %				
	+1.4 (mm)	+0.84 (mm)	+0.59 (mm)	+0.42 (mm)	-0.42 (mm)
1	2.4	1.9	2.3	10.1	83.3
2	1.9	2.0	1.7	9.6	84.8
3	4.6	2.0	2.3	11.2	79.9
4	0.4	2.2	8.8	21.3	67.4
5	2.8	1.5	1.5	9.2	85.0
6	3.0	3.0	3.0	9.1	81.8
7	0.9	0.9	0.9	6.1	91.3
8	1.9	1.3	1.9	10.3	84.5

TABLE 24
Size Analysis of Spent
Shale from the Retort

Sample No.	Weight, %					
	+8 (mm)	+2.8 (mm)	+1.4 (mm)	+0.71 (mm)	+0.15 (mm)	-0.15 (mm)
1	20.3	63.6	13.6	1.4	0.3	0.8
2	18.7	61.5	15.7	2.7	0.6	0.7
3	8.5	7.7	21.6	10.0	11.6	10.7
4	7.9	51.0	14.5	8.4	12.2	6.0
5	16.1	62.8	17.6	1.8	0.5	1.2
6	22.7	58.5	13.5	3.4	1.3	0.5
7	5.7	41.0	15.6	10.5	15.7	11.5
8	29.1	50.2	16.4	2.4	0.9	1.2

APPENDIX E

Shale Oil Distillation Data

TABLE 25

Distillation Characteristics of Shale Oil

Volume % Distilled	Sample #1 °C	Sample #2 °C	Sample #3 °C	Sample #4 °C	Sample #5 °C
0.5	86	85	81	81	83
5	169	167	157	160	159
10	203	198	195	195	194
15	230	225	221	220	218
20	252	248	245	243	240
25	271	268	267	265	262
30	290	287	287	285	282
35	309	305	305	303	300
40	327	323	324	321	316
45	344	342	342	341	335
50	362	358	360	358	353
55	379	376	377	376	370
60	395	392	394	393	387
65	412	410	411	411	404
70	429	426	428	428	422
75	447	444	446	446	440
80	467	463	465	466	460
85	491	486	488	489	482
90	522	514	517	518	511

Distillation Characteristics of Shale Oil

Volume % Distilled	Sample #6 °C	Sample #7 °C	Sample #8 °C	Knockout #1 °C	Knockout #2 °C
0.5	83	93	109	253	203
5	157	171	174	410	395
10	193	201	206	455	445
15	216	226	230	481	470
20	237	248	251	502	489
25	258	268	269	521	504
30	278	286	287		518
35	297	303	305		532
40	314	321	322		
45	330	339	340		
50	348	356	357		
55	365	373	374		
60	381	390	391		
65	399	407	409		
70	416	424	426		
75	434	442	445		
80	453	461	465		
85	475	484	489		
90	503	514	521		

APPENDIX F

Calculation of Combustor Flue Gas Flow Rate

CALCULATION OF COMBUSTOR FLUE GAS FLOW RATE

Flue gas Composition (Table 6)	CO_2	14.6%	
	O_2	3.2	
	CO	0.481	
	SO_2	0.3634	
	NO_x	0.0208	
	N_2	81.34	(by diff.)
		100.0	

Average Molecular Wt

$(0.146 \times 44.01) + (0.032 \times 32) + 481 \times 10^{-6} \times 28.0)$
$+ (36 \times 10^{-6} \times 64) + (208 \times 10^{-6} \times 46\ (NO_2)) + (0.813 \times 28)$
$= 30.24$

Flow Rate of Dry Flue Gas Based on Nitrogen Balance

N_2 in air (feed)=total air feed ratexwt fraction, N_2 in air

$= 99.4 \times 0.767$
76.24 kg/hr

Moles N_2 IN = moles N_2 OUT

$\frac{76.24}{28}$ = moles dry flue gas x mole fraction N_2 in flue gas

= moles dry flue gas x 0.8134

$\therefore$ Moles of dry flue gas $= \frac{76.24}{28 \times 0.8134} = 3.248$ kg mole/hr

$= 3.348$

wt dry flue gas $= 3.348 \times 30.24$
$= 101.24$ kg/hr

Moisture in flue gases

coal composition (Table 2) = 0.45% H_2O (moisture)
4.72% H_2

feed rate (Table 5) = 8.5 kg/hr

rate of water formation from moisture and combustion (based on coal only) $= 8.5 \left(\frac{0.45}{100} + \frac{4.72 \times 18}{100 \times 2}\right)$

= 3.65 kg/hr

Total weight of flue gases

wt of wet flue gas = 3.65 + 101.24
= 104.89 kg/hr

In Table 5 the flue gas flow rate was calculated by overall mass balance as follows:

wt flue gas = TOTAL IN - TOTAL OUT
= 120.6* - (6.22 + 5.21 + 1.19)**
= 107.98 kg/hr

* Table 5 - sum of coal, air and spent shale entering

** Table 5 - sum of bed, secondary cyclone and baghouse discharges exiting

APPENDIX G

Method of Calculating

Combustor Air and Retort Gas Flow Rates

Method of Calculating Combustor Air and Retort Gas Flow Rates

Orifice Plate 0.6" in 1.5" pipe

$$\text{flowrate} = K \sqrt{\Delta \frac{PxPxMW}{T}} \quad \text{lb/min}$$

where K = oriface constant = 0.4053

Δ P = pressure drop (inches water)

P = Absolute pressure (Psig)

MW = Molecular weight of air

T = absolute temperature (^{o}R)

at $70^{o}F$ i.e. 530R

$$\text{flowrate of air} = 2.578 \sqrt{\Delta PxP} \quad \text{kgm/hr}$$

Rotameters

$$\text{flowrate} = 0.1734 \times \sqrt{P/24.7} \quad \text{kgm/hr}$$

where X = % flow

P = absolute pressure (Psig)

Combustor Air Flow (example calculation)

Air flow

primary air $= 2.578 \sqrt{\Delta P \times P}$

$= 2.578 \sqrt{45 \times (4.2 + 14.7)}$

$= 75.18$ kg/hr

rotameters $= 0.1734 \times \sqrt{P/24.7}$

aeration 1st $= 0.1734 \times 39 \sqrt{14.7 + 3.5/24.7} = 5.8$ kg/hr

lift 2nd $= 0.1734 \times 32 \sqrt{14.7 + 2.4/24.7} = 4.62$ "

feeder 3rd = $= 9.5$ "

sec. cyclone 4th = $= \underline{3.56}$ "

TOTAL $= 98.66$ "*

From Table 18C:-

Primary	Rotameters
45 = av. diff. pressure	39 = av % flow aeration
4.2 = av. G. pressure	3.5 = av G. pressure aeration
etc.	32 = av % of flow lift
	2.4 = av G. pressure lift
	etc.

* Note value given in Table 5 is 99.4 kg/hr

Retort gas flow rates (example calculation)

Orifice equation for retort

$$\text{flow rate} = K\sqrt{\frac{\Delta P \times P \times MW}{T}} \quad \text{lb/min}$$

K = orifice constant
= 0.5082 total recycle
= 0.5084 middle recycle
= 0.2537 bottom recycle

ΔP = press drop (inches water)
P = absolute or Guage press (Psig)
MW = Molecular wt
T = Temperature (Rankine) ($F^o + 460^o$)

Use MW = 26.3 (table 16 av 26.43 + 26.18)

ΔP = 22.7" (av. diff pressure total gas flow table 19B

P = 4.6 psig (av. total gas flow G. press table 19B
= 4.6 + 14.7 = 19.3 Psig

T = 88^oC (temp total gas table 19B)
= $190.4^oF = 650.4^oR$

$$\text{Flow rate} = 0.5082\sqrt{\frac{22.7 \times 19.3 \times 26.3}{650.4}}$$

= 2.139 lb/min

$$= \frac{2.139 \times 60}{2.205} = 58.2 \text{ kg/hr}^*$$

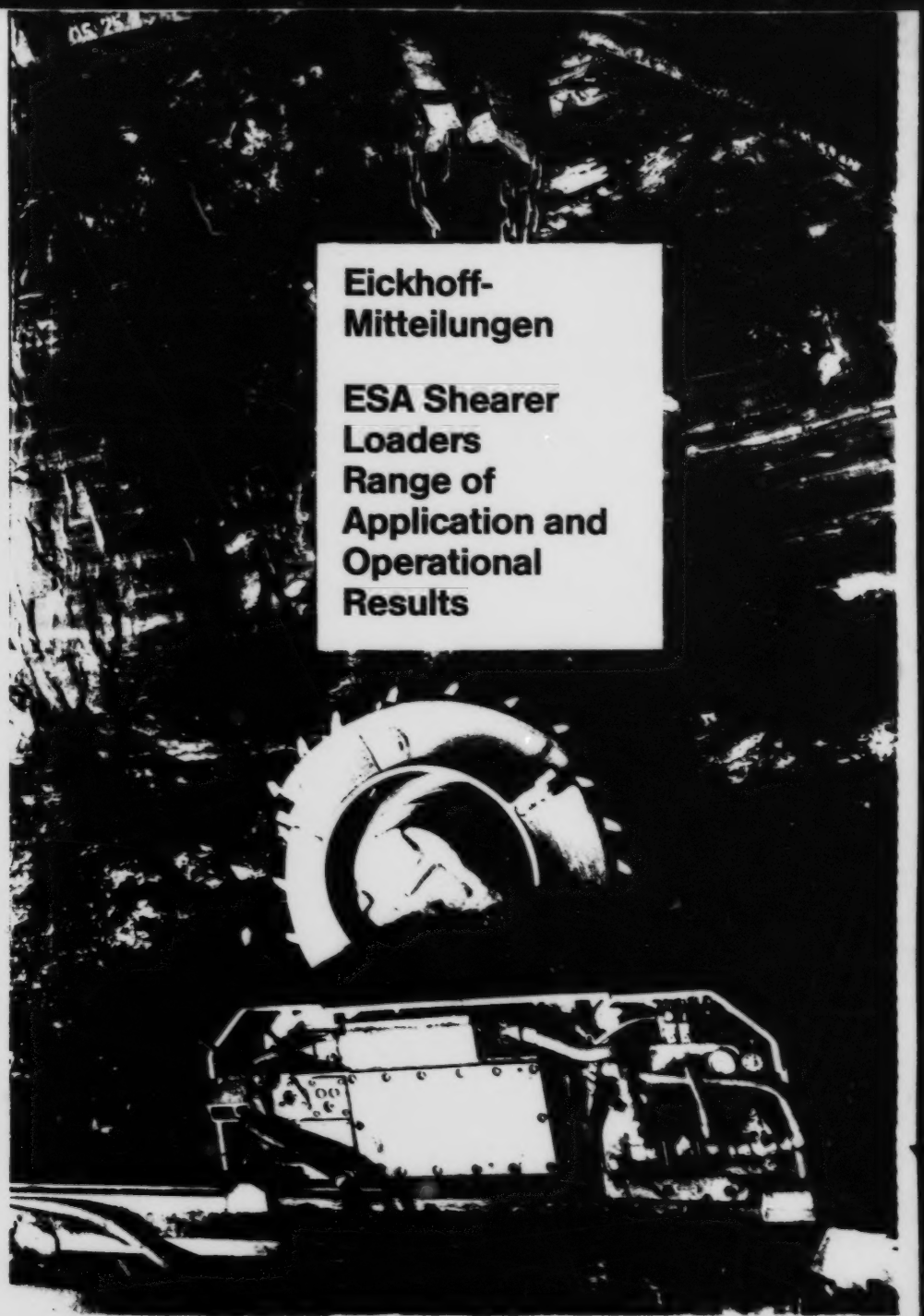
Value is an "average" that differs from the 3 values given in Tables 12-14.

APPENDIX H

Abbreviations Used in Tables and Figures

Abbreviations Used in Tables and Figures

cond	condenser
C W	cold water
cyc dis	cyclone discharge
diff press	differential pressure
esp	electrostatic precipitator
G press	gauge pressure
ht exch in	heat exchanger in
ht exch out	heat exchanger out
in water	inches water
sec cyc	secondary cyclone
sec cyclone rec	secondary cyclone recycle
sec cyc discharge	secondary cyclone discharge
stnp bot	standpipe bottom
stnp mid	standpipe middle
stnp dis	standpipe discharge
TC	thermocouple



Eickhoff-Mitteilungen

ESA Shearer Loaders Range of Application and Operational Results

Eickhoff-Mitteilungen
54th year · issue 1 · January 1987

ESA Shearer Loaders are used as stablehole, face opening, shortwall, pillar extraction and heading machines.

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More than 80 machines of the ESA-60 and ESA-150 series are operating successfully worldwide.

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1. Introduction

Since the first Eickhoff stable hole and face opening machine was installed at Zolder Colliery in Belgium in 1979, this type of machine has demonstrated much success in a variety of underground operations.

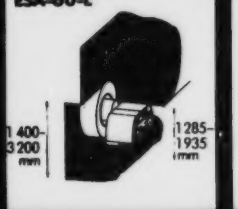
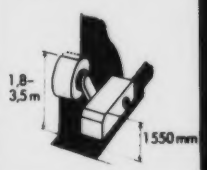
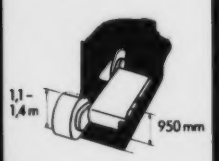
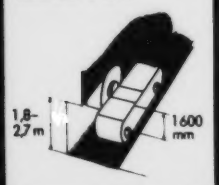
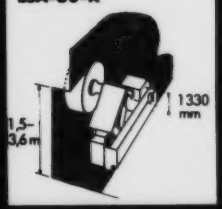
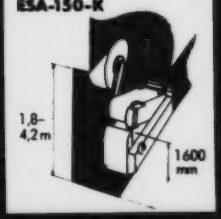
The first installations only served for the mechanization of face opening and stablehole operations in order to reach higher advancing rates than could be achieved with conventional methods on the one hand, and on the other to contribute to the further humanization of the working environment.

As a winning machine on shortwall faces the ESA successfully accomplishes its task with an installed motor power of both 60 kW and 150 kW.

The ESA shearers with tiltable machine body can be used for cutting the desired roadway cross-section independent of the mining method applied on the face. In the case of in-line roadway headings the machines can be mounted on the face conveyor and in the case of advanced headings they can be operated in conjunction with special furnishings. In any case they contribute largely to solving the problems encountered at the face ends.

Following the presentation of the ESA system various installations at home and abroad are described.

ESA Machines

	60 Series	150 Series
basic version	**ESA-60-L**  	**ESA-150-L** 
double ranging or fixed drums	**ESA-60-2L / -2**  	
version with two haulage units	**ESA-60-L-2W**  	
articulated ranging arm	**ESA-60-K**  	**ESA-150-K** 

The entire ESA series – with the exception of the ESA-60/60-2 – are equipped with a pivoting gear, so that the ranging arm can be turned through 360°.

2. Description of the machine

The ESA consists of the following main components:

cutting drum and ranging arm
ranging arm bearing
electric motor
pump drive
travelling and pivoting drives
underframe

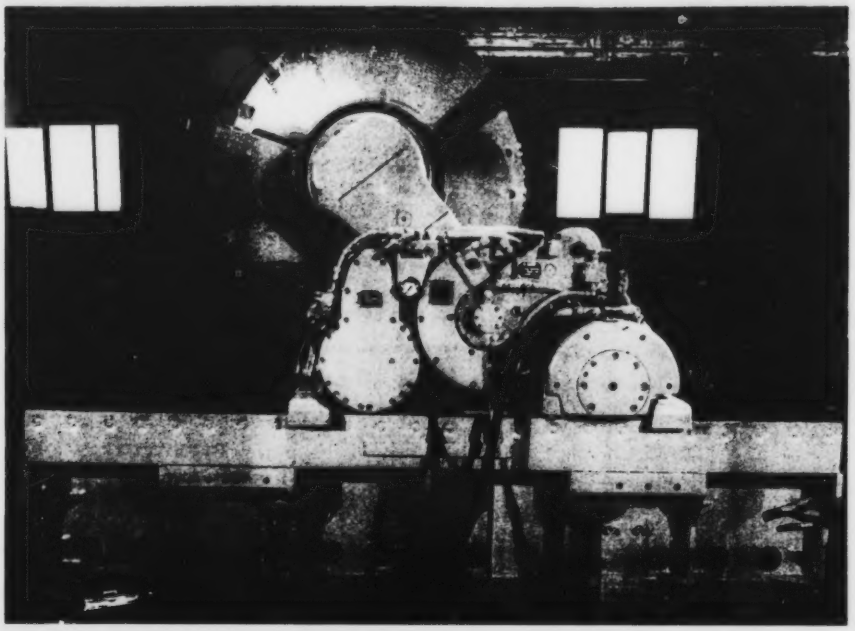
The components of the machine are supported on the underframe which slides with the shoes bolted to it and the face side trap guiding arrangement on the accessories thus guiding the machine.

The centrally arranged electric motor drives the drum and the pivoting gearing of the ranging arm as well as the haulage unit. The ESA-60-2 is an exception in as much as each of the laterally arranged electric motors drives a cutting drum and one of them also drives the hydraulic pump of the haulage unit.

Together with the haulage unit the rack bar forms the motion element. The machine is hauled by the hydraulic haulage unit. The drive wheel of the machine is positioned on the goaf side and is part of the Eicotrack system by means of which the machine slides along the face.

The ESA principle which enables the machine to clear its way has been realized differently for the various versions. All versions, with the exception of the ESA-60/60-2, are equipped with a ranging gearbox which allows for turning the ranging arm through 360°. The ESA-60-L can pivot the ranging arm both along the roof and along the floor as the special accessories can be adapted to such operating conditions. In the case of the ESA-150-L the ranging arm can be pivoted either along the roof or along the floor. The ESA-K series can raise the machine body by means of a hydraulic cylinder which is mounted to the middle of the machine. The maximum tilting angle of the machine body in relation to the underframe amounts to 65° for the ESA-60-K and 53°20' for the ESA-150-K. Lifting of the machine allows for reaching the maximum cutting height and enables the ranging arm to be pivoted along the floor.

The cutting range of the ESA-60-2 depends directly on the diameter of the cutting drum. If the cutting range must be increased or reduced this can be achieved by displacing the ranging arm on the connection flange of the bearing casing and by selecting an appropriate cutting drum in order to reach the required height.



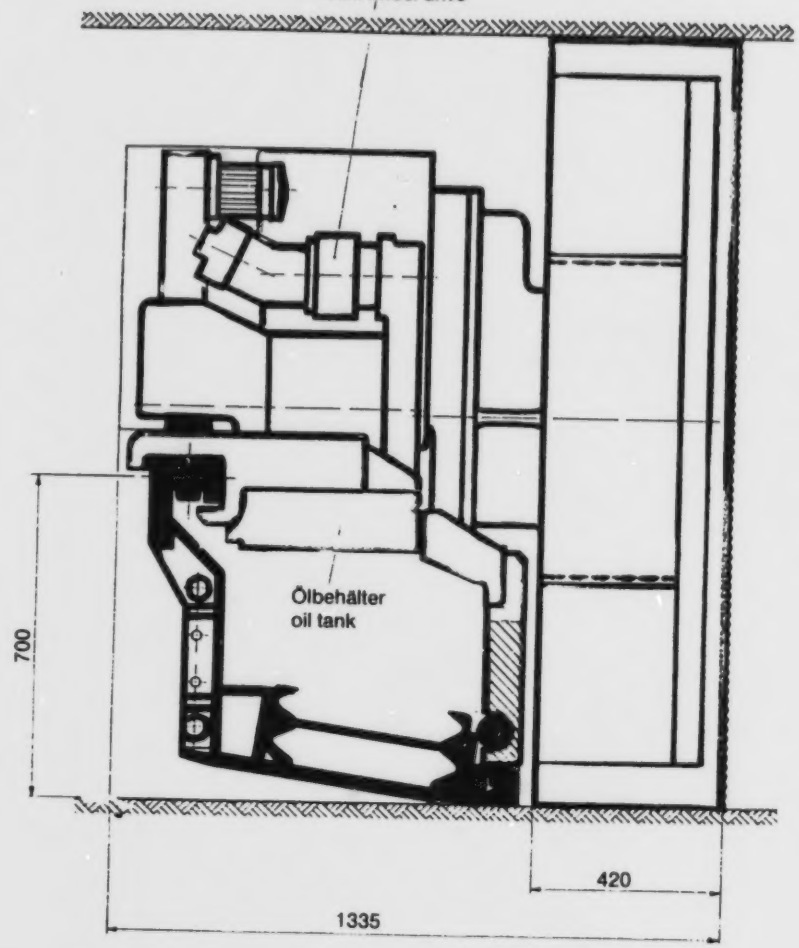
Eickhoff Stable Hole and Face Opening Machine ESA-60-L

Machine height	approx. 1285 – 1935 mm
Ranging arm length	250, 500, 750 mm
Cutting drum diameter	1400 – 1800 mm
Cutting depth	420 – 625 mm
Cutting range	3100 mm
Drum speed	32 r.p.m.
Admissible continuous output torque	18 kNm
Total installed continuous motor power	60 kW
Weight	approx. 5,5 t – 9 t
Nominal voltage	500 V, 1000 V
Frequency	50 Hz
suitable for the face conveyor	EKF-O
Haulage system	Eicotrack

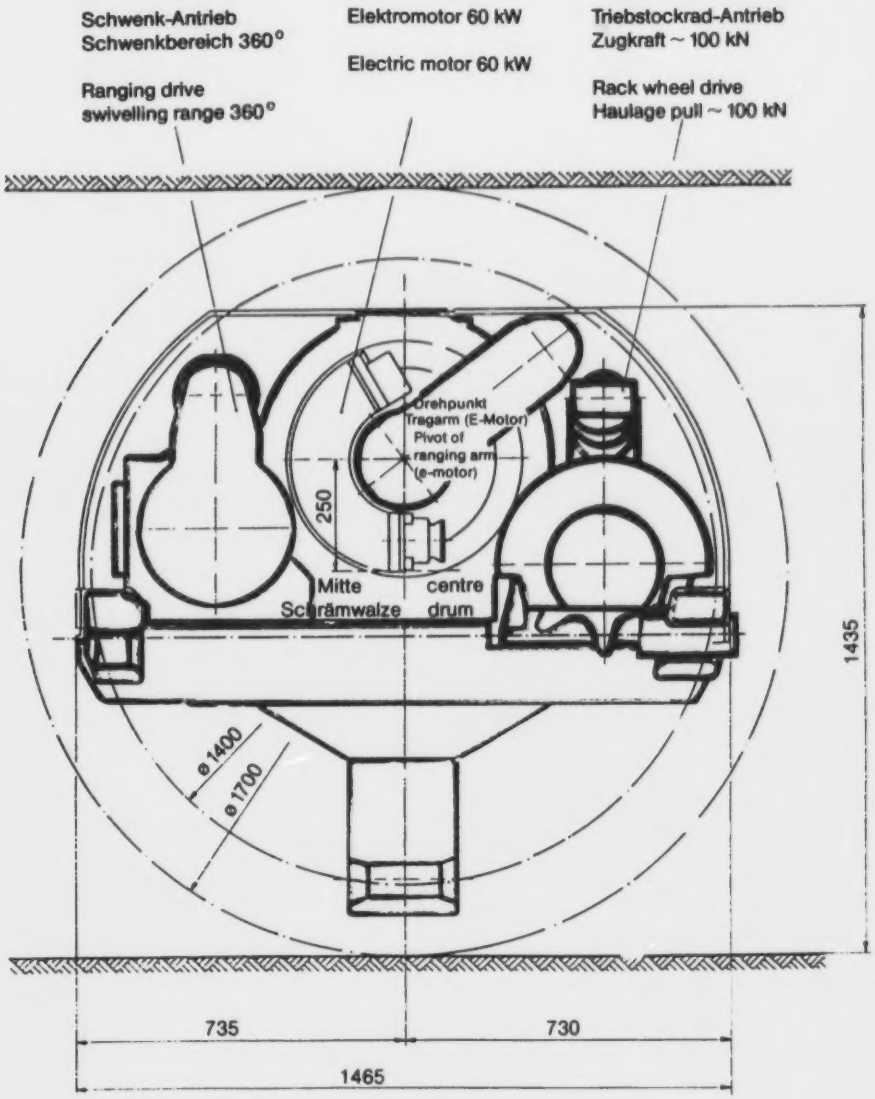
The haulage pull of 110 kN is reached at a travelling speed of 0 to 180 m/h.

Hydraulik-Motor
mit Planetengetriebe
für Triebstockrad-Antrieb

Hydraulic motor with
planetary gearing for
rack wheel drive



Fahr- und Schwenkgeschwindigkeit ~ 3 m/min
travelling and swivelling speed ~ 3 m/min



Stall- und Aufhauenmaschine ESA-60-L

Stable Hole and Face Opening Machine ESA-60-L

Each of the laterally arranged elec one drum. One of the motors also haulage unit.

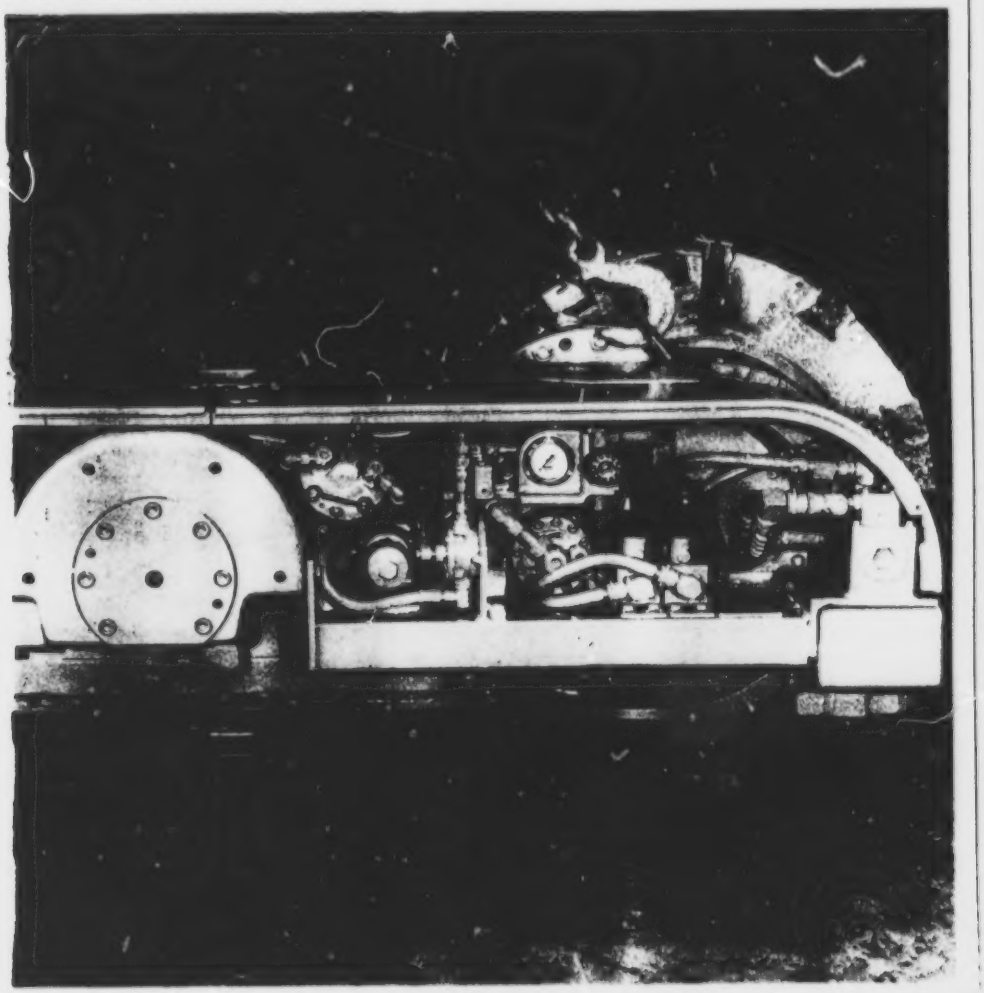
Eickhoff Stable Hole and Face Opening Machine ESA-60-2

Machine height (approx.)	980 mm
Ranging arm length	250 mm
Cutting drum diameter	1300 / 1500 mm
Cutting depth	420 mm
Undercut	20[1] / 22[2] mm
Drum speed	32 r.p.m.
Admissible continuous output torque	18 kNm
Total installed continuous motor power	60 kW
Weight (approx.)	6,7 t
Nominal voltage	1000 V
Frequency	50 Hz
Maximum gradient	level
Haulage system	Eicotrack

The maximum haulage pull of 110 kN is reached at a travelling speed of 0 to 180 m/h.

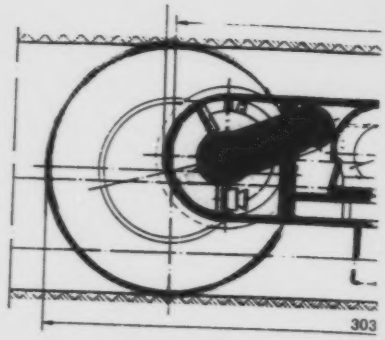
[1] at a drum diameter of 1300 mm
[2] at a drum diameter of 1500 mm

tric motors of the ESA-60-2 drives
drives the hydraulic pump of the



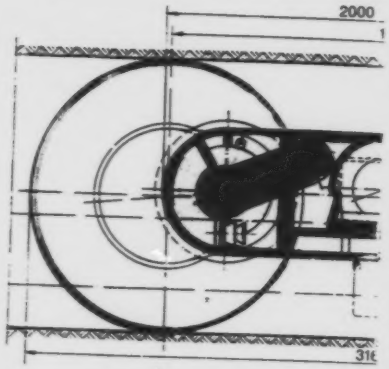
ESA-60-2 mit starren Walzentragarmen
mit Walzen ø 1100 mm

with rigid drum supporting arms
with 1100 mm drum diameter



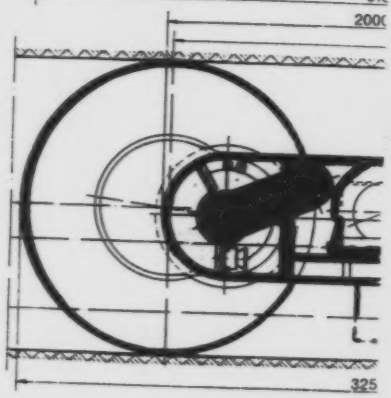
1. Alternative
mit Walzen ø 1200 mm

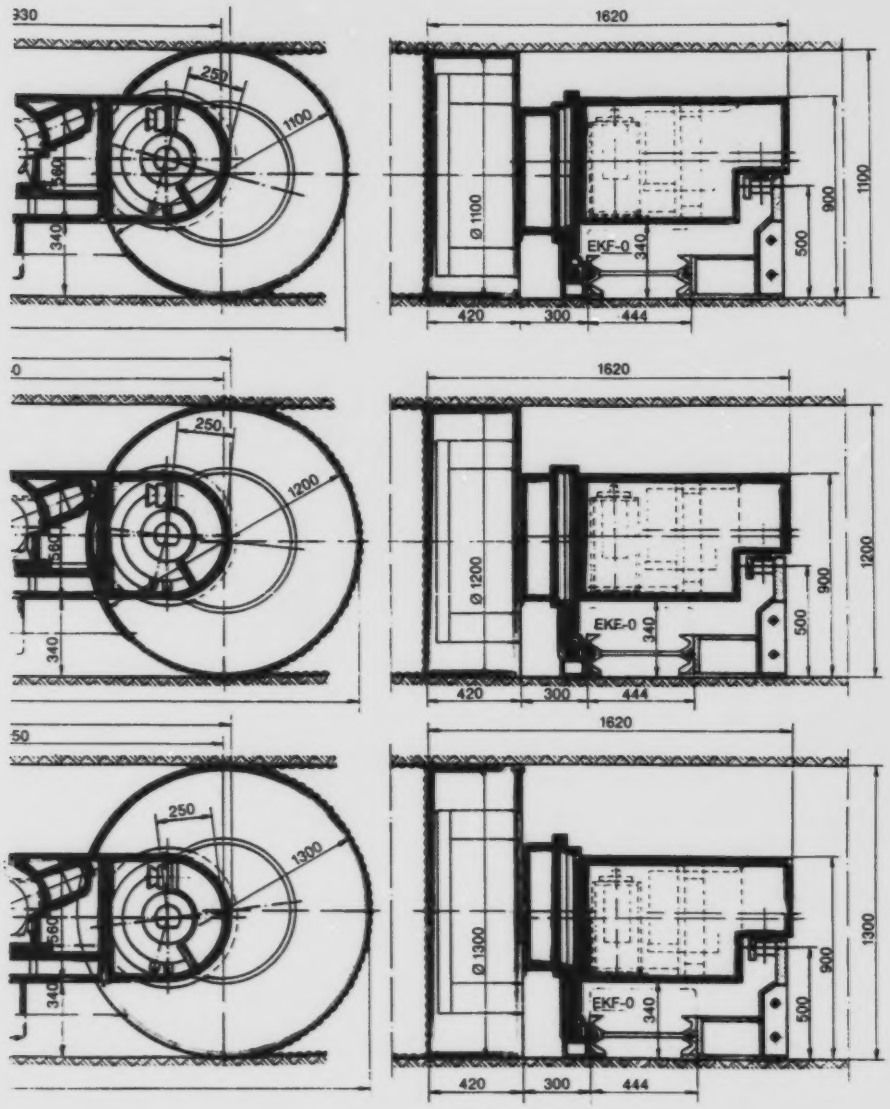
1st alternative
with 1200 mm drum diameter

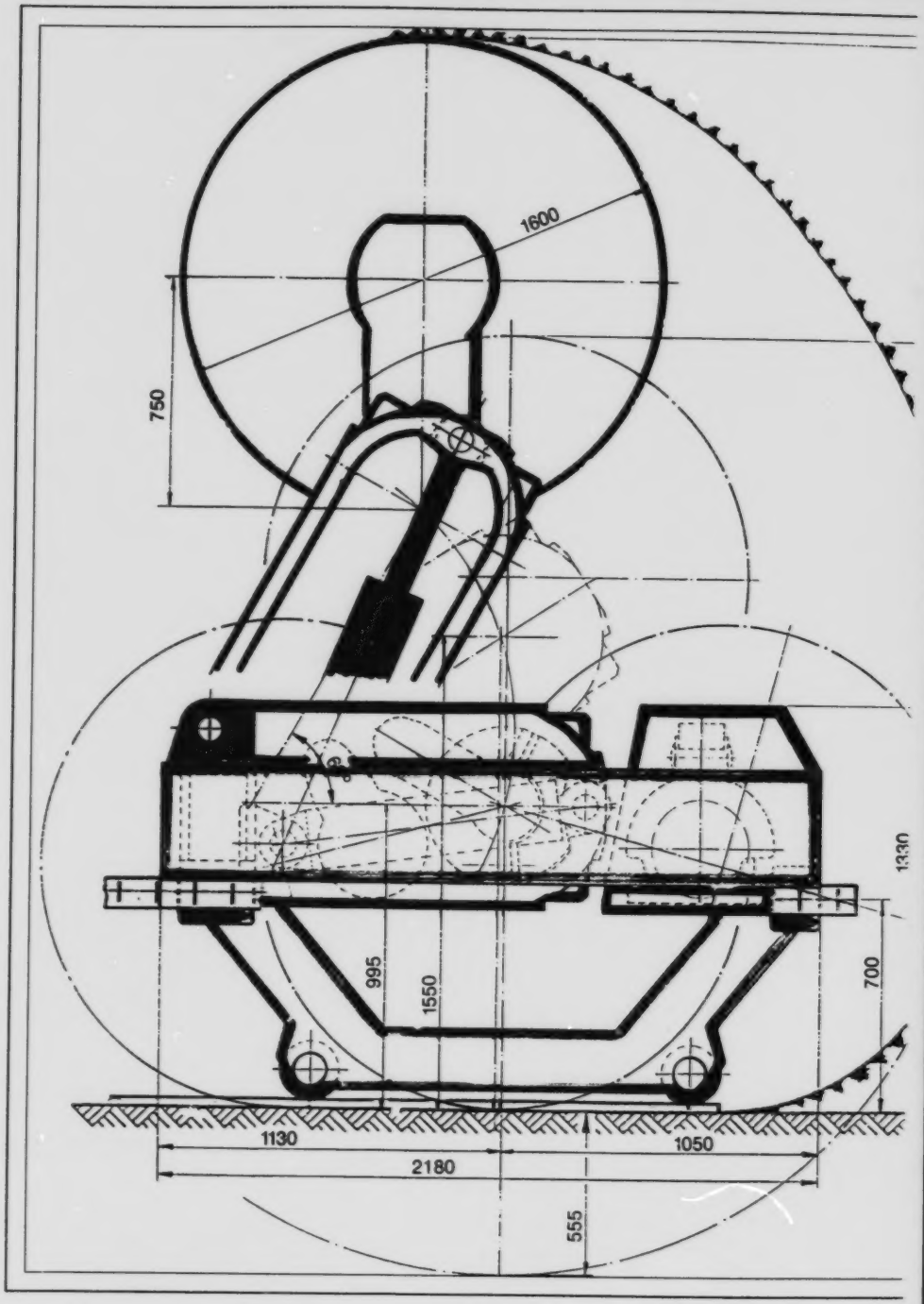


2. Alternative
mit Walzen ø 1300 mm

2nd alternative
with 1300 mm drum diameter




330
250
1100
560
340
1620
Ø 1100
EKF-0
340
500
900
1100
420
300
444
0
250
1200
560
340
1620
Ø 1200
EKE-0
340
500
900
1200
420
300
444
50
250
1300
560
340
1620
Ø 1300
EKF-0
340
500
900
1300
420
300
444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750
995
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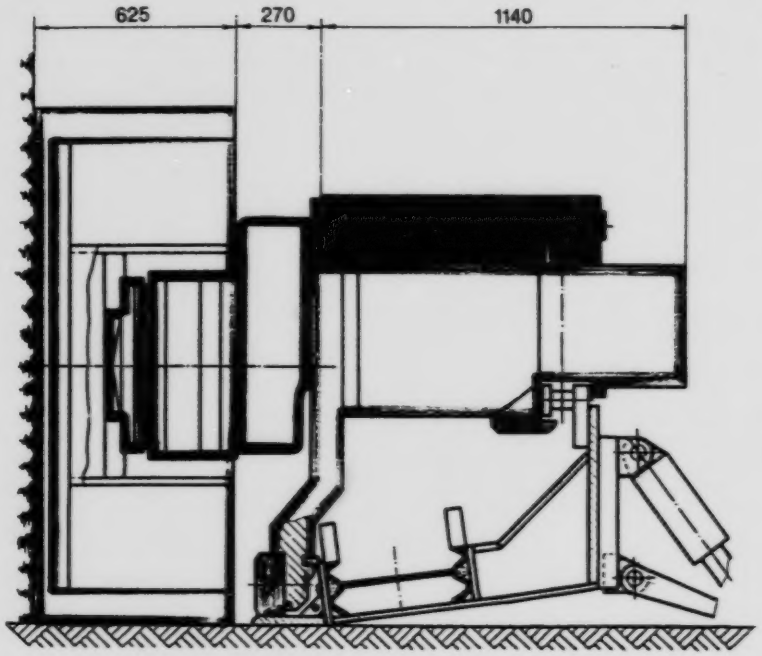
555



Eickhoff Face End and Heading Machine ESA-60-K

Machine height (approx.)	1300 mm
Ranging arm length	750 mm
Cutting drum diameter	1500 / 1600 mm
Cutting depth	625 mm
Cutting height	3620 mm
Undercut	555 mm
Drum speed	32 r.p.m.
Admissible continuous output torque	18 kNm
Total installed continuous motor power	60 kW
Weight (approx.)	12 t

Nominal voltage	500 V, 1000 V
Frequency	50 Hz
Gradient	level
suitable for the face conveyor	EKF-O, EKF-2, EKF-3
Haulage system	Eicotrack

The maximum haulage pull of 110 kN is reached at a travelling speed of 0 to 288 m/h.

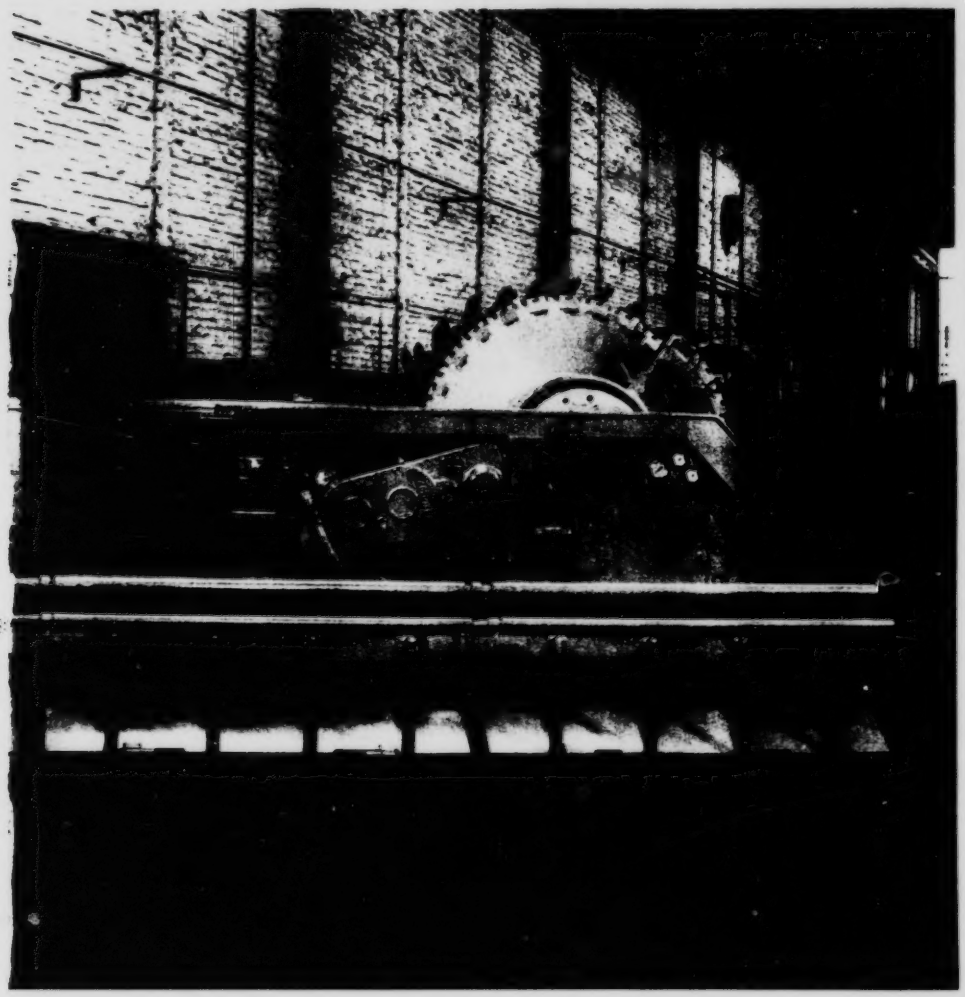
The Shortwall ESA-150-L is desigr
arm can be swivelled either along t

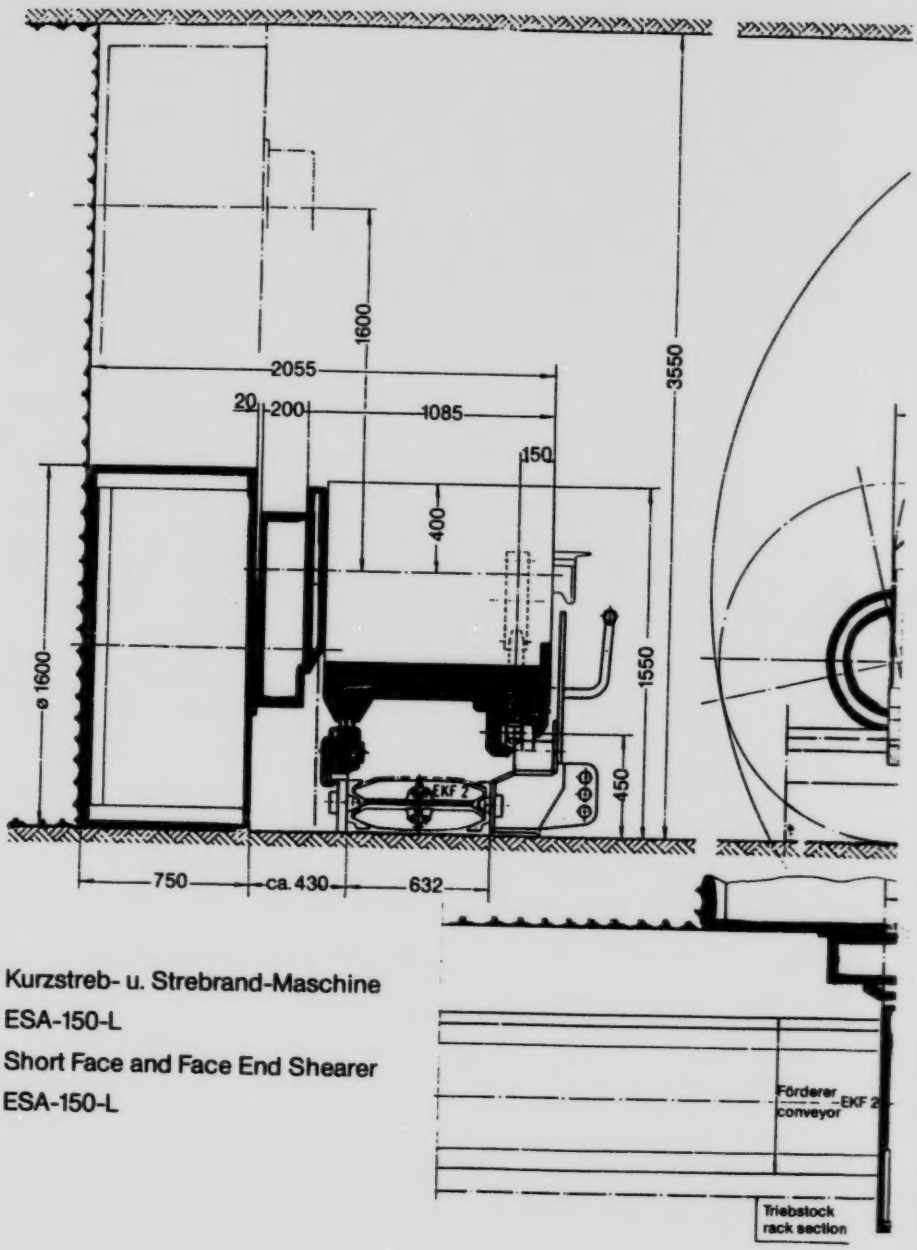
Eickhoff Shortwall Machine ESA-150-L

Machine height	1500 – 1600 mm
Ranging arm length	1050, 1337, 1600 mm
Cutting drum diameter	1600 mm
Cutting depth	625, 800 mm
Cutting height	max. 3600 mm
Undercut	approx. 300 mm
Drum speed	32, 39 r.p.m.
Admissible continuous output torque	37, 45 kNm
Total installed continuous motor power	150 kW
Weight (approx)	16 t
Nominal voltage	1000 V
Frequency	50 Hz
Gradient	level
suitable for the face conveyor	EKF-2
Haulage system	Eicotrack

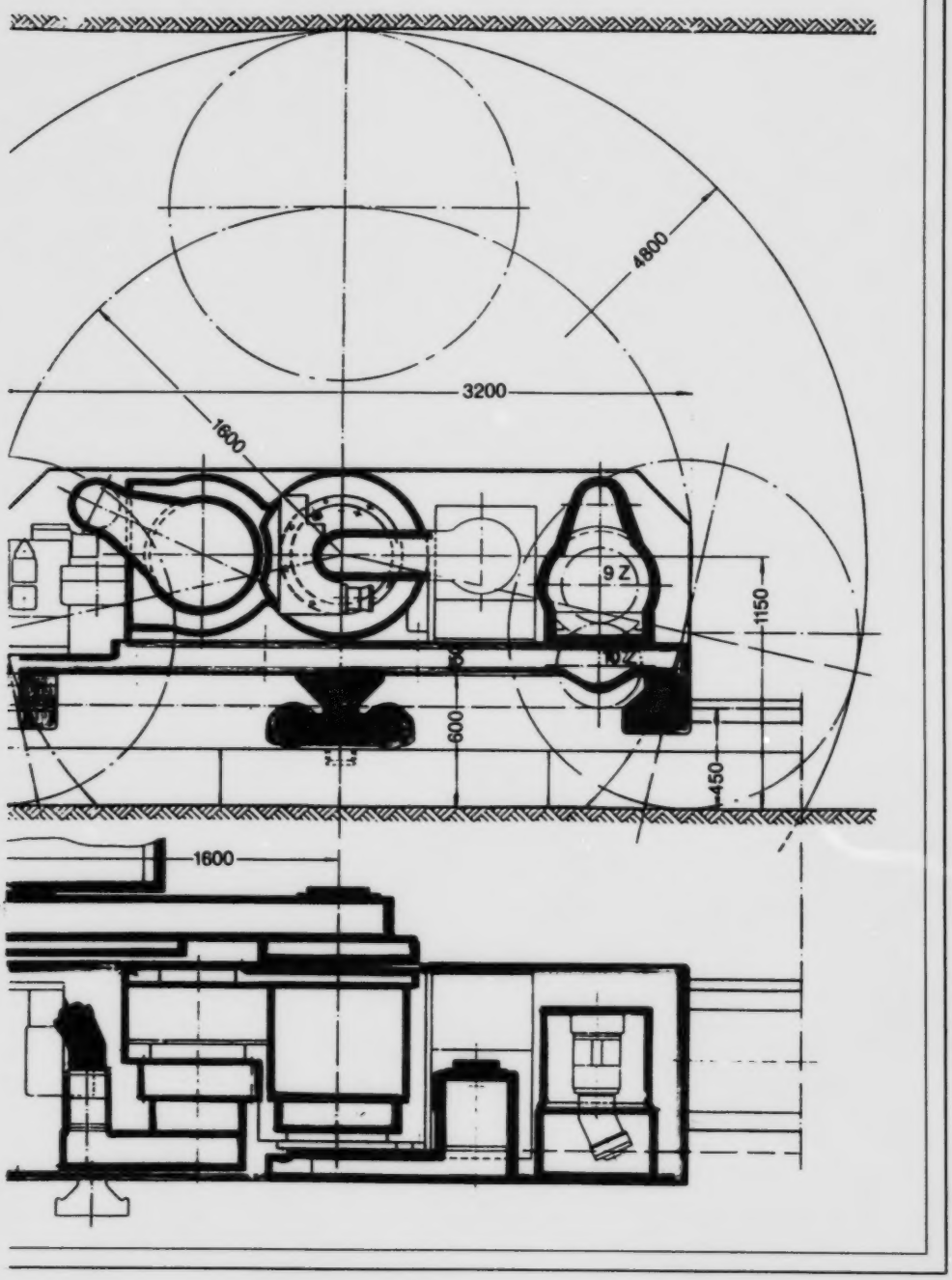
The maximum haulage pull of 250 kN is reached at a travelling speed of 0 to 372 m/h.

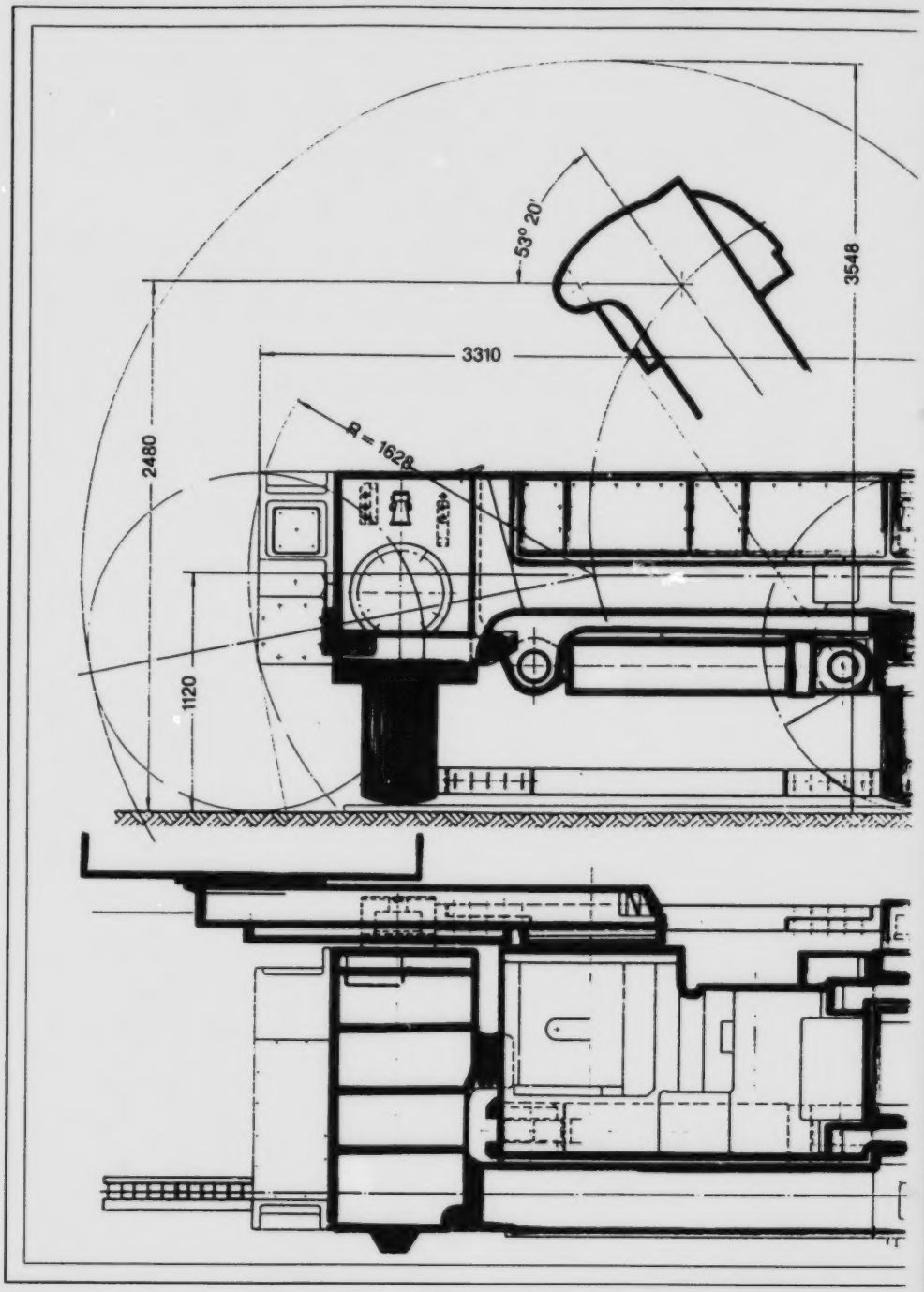
ed in such a way that the ranging he floor or along the roof.

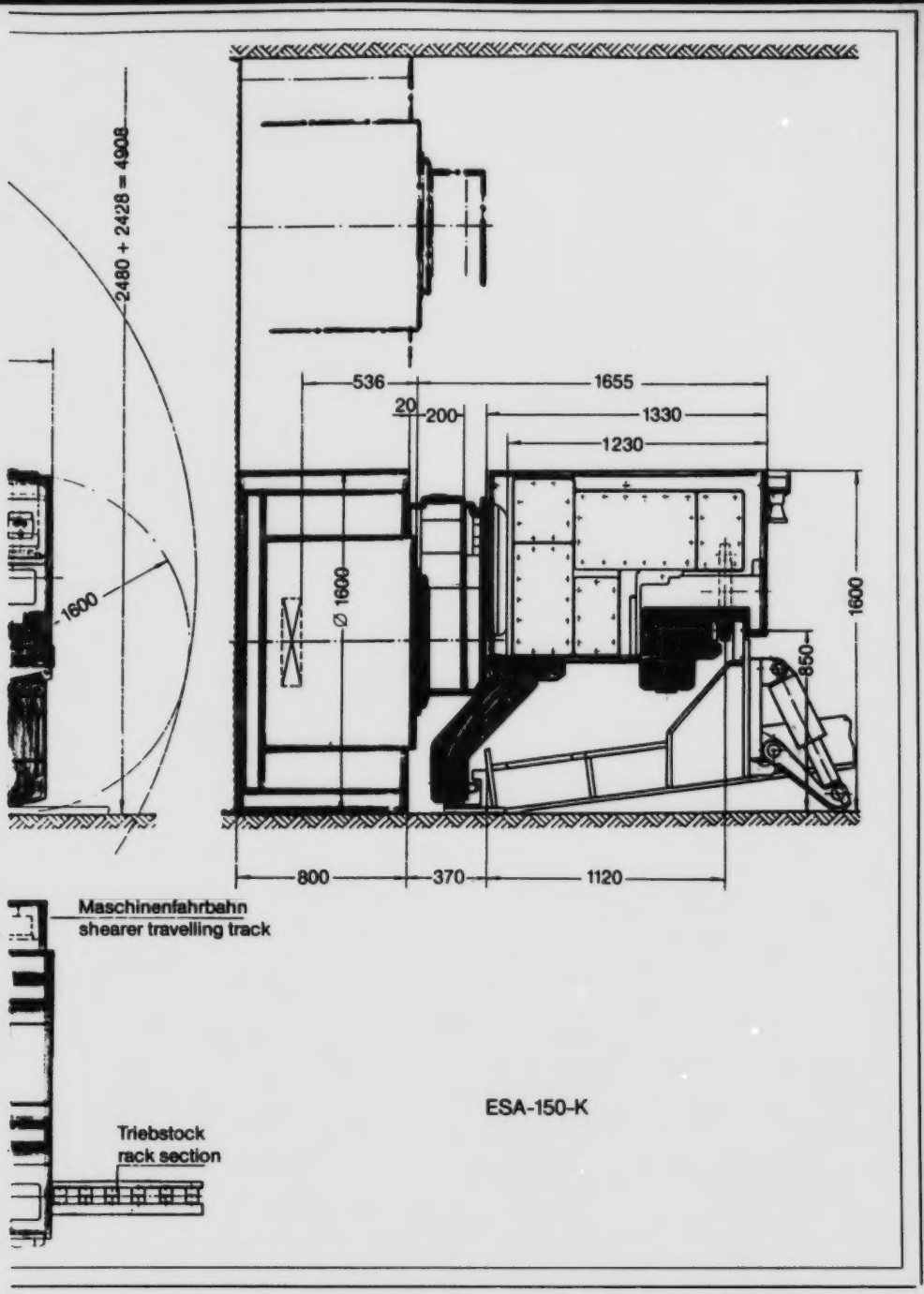




Kurzstreb- u. Strebrand-Maschine
ESA-150-L
Short Face and Face End Shearer
ESA-150-L


4800
3200
1600
1150
600
450
1600


53° 20'
3548
3310
R = 1628
2480
1120


2480 + 2428 = 4908
1600
536
1655
20
200
1330
1230
Ø 1600
1600
850
800
370
1120
Maschinenfahrbahn
shearer travelling track
Triebstock
rack section

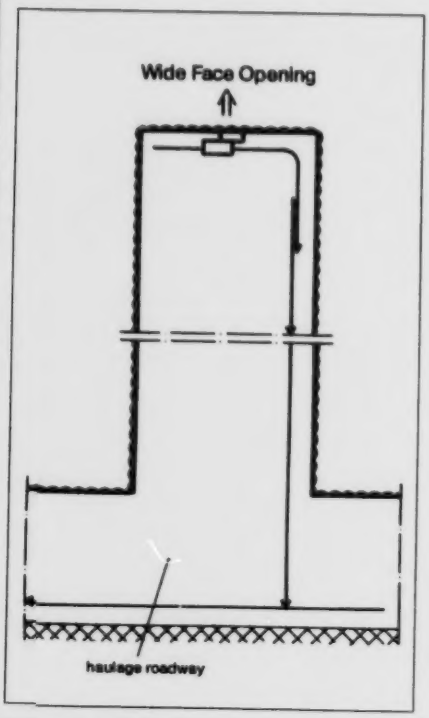
ESA-150-K

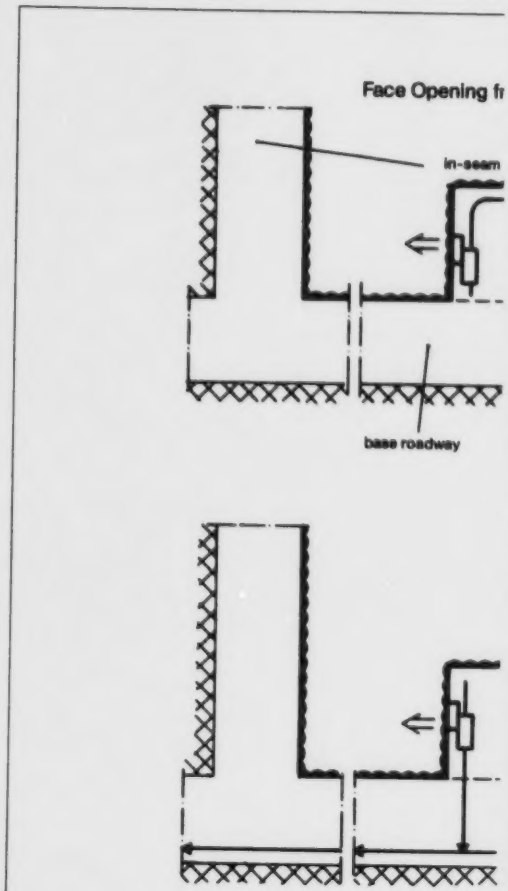
The mechanical face opening tech cavities of small width are driven c a relatively short period of time.

3. Operating Ranges

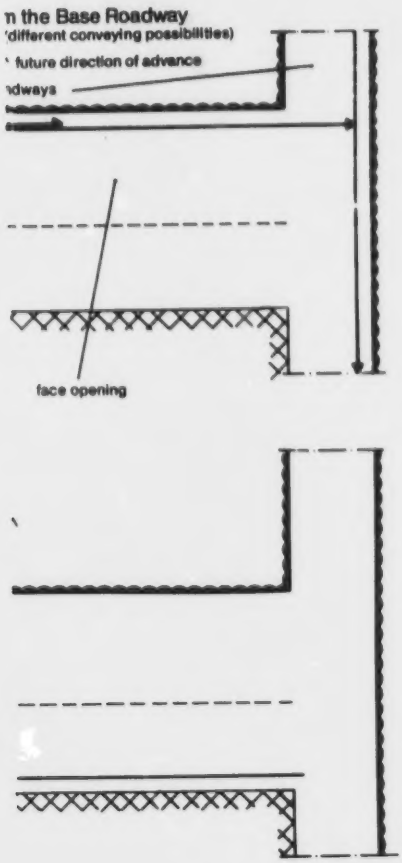
3.1 Face Opening to the Dip and to the Rise

The mechanical face opening technique is devised to create rectangular cavities of small width and up to a length of 250 m, in some cases under a friable roof and on a soft floor, within a relatively short period of time.





nique requires that rectangular ver a length of up to 250 m within



3.2 Face End and Heading Operations

High face advancing speeds can only be reached if the face end technique is adapted to the face operations in an optimum way. With regard to location and time and the dependence of the roadway heading to the position of the face the face end area can be classified as:

- in-line roadways
- lagging roadways
- advanced roadways.

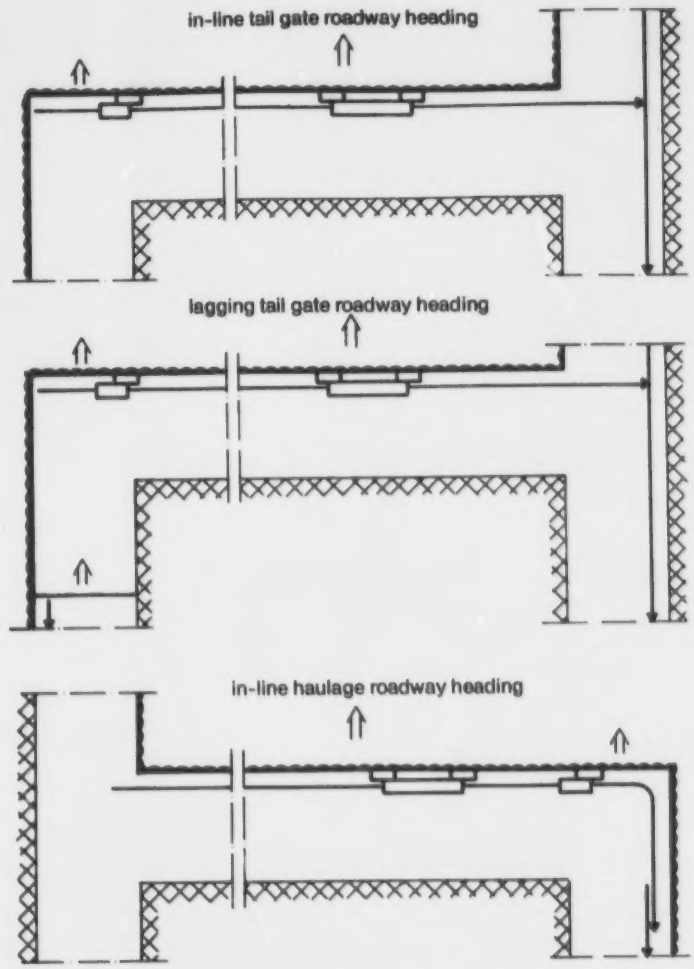
The problems encountered in every particular case will be outlined elsewhere.

All three of the categories mentioned lend themselves for the installation of the ESA system which is partly not only used purely for heading operations, but also as an efficient winning machine and as a means of simplifying the mining cycle of the face machine (stablehole machine).

In-line and lagging in-seam roadways:

An essential feature is that the ESA is mounted on the face furnishings. The accessories otherwise required are omitted. Depending on its type and the seam thickness involved the machine can perform the following functions:

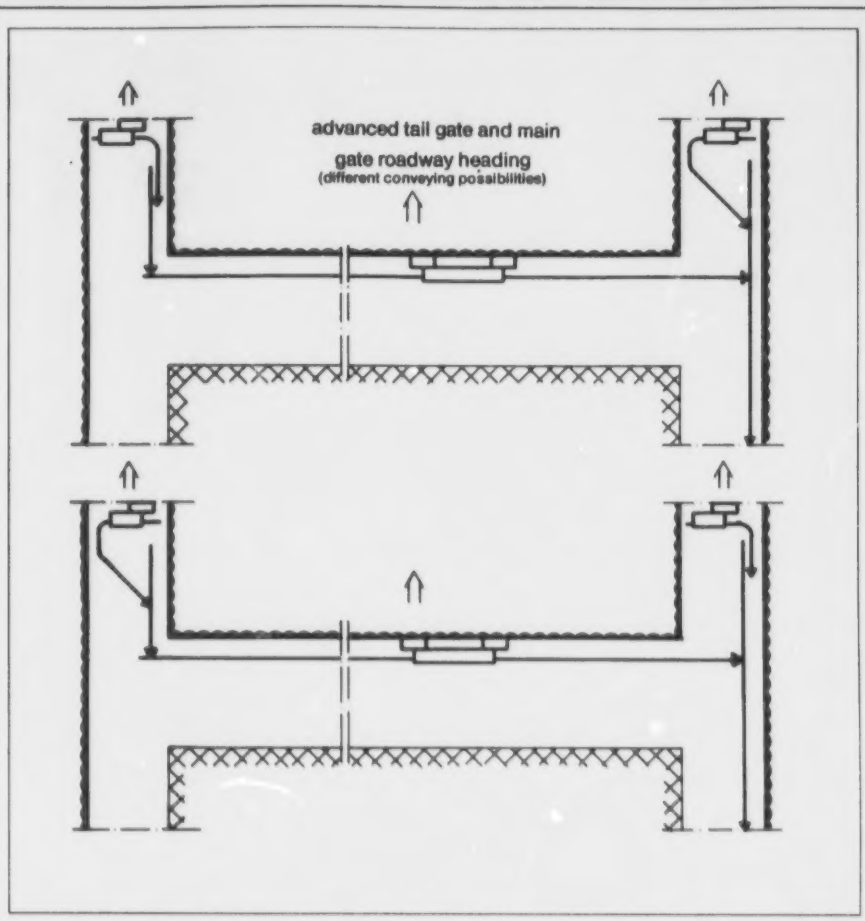
- partial extraction of the roadway cross-section as well as elimination of shunting on shearer faces or of the stablehole on plough faces;
- extraction of the whole roadway cross-section (in thicker seams there is also the possibility to elimi-



In-line and lagging roadway headings with the ESA

nate shunting or to cut the stablehole for the plough).

The illustration shows the possible applications in the tail gate and in the main gate.



Advanced roadway headings with the ESA

Advanced Headings:

In advanced in-seam roadway headings the machine is mounted on its own accessories. The conveyor can be arranged in different ways.

The distance from the face generally amounts to at least 15 – 20 m to enable working in an area outside of the pressure zone preceding the face.

The illustration shows known possibilities gained from experience.

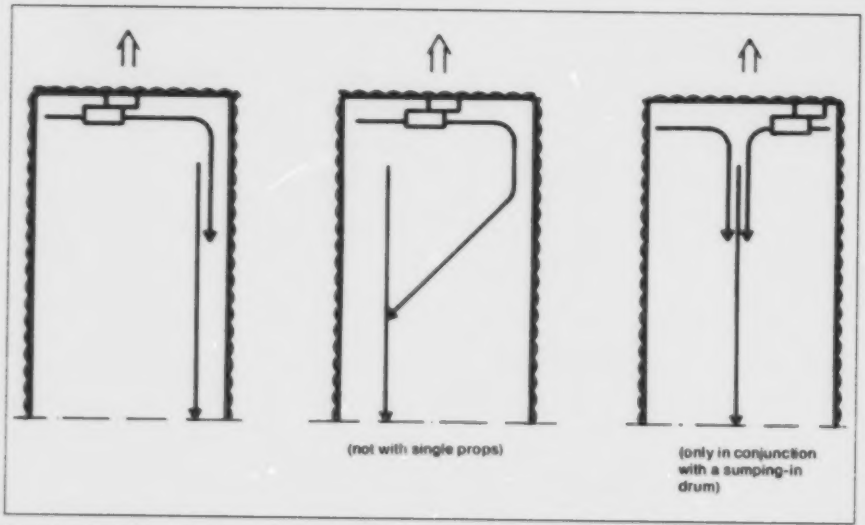
Roadway headings not depending on the normal mining operation, pillar roadways:

Further application areas are roadway headings which are independent of the mining operation and which can be termed as pure development roadways. The drivage of pillar roadways in room-and-pillar workings can also be mechanized by the ESA system. Although the cross-section of such workings and the arrangement of the ESA system are very

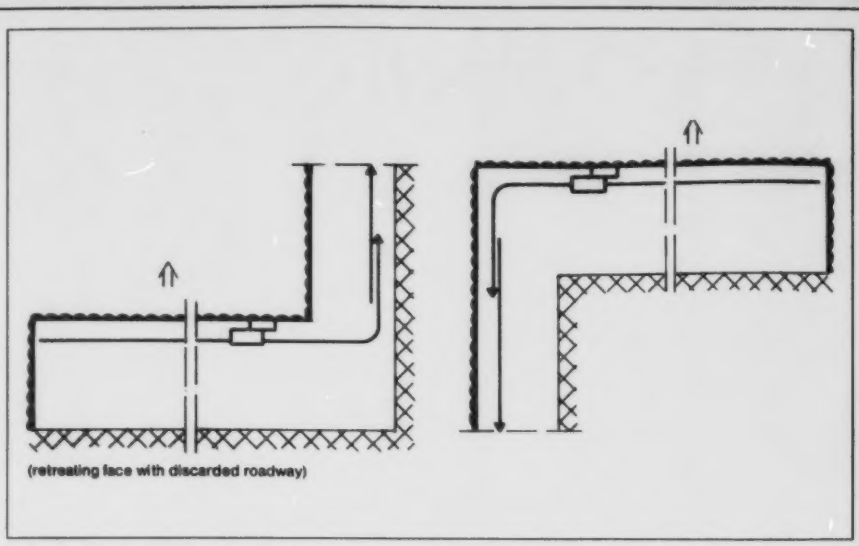
In in-line, lagging and advanced roadway headings the ESA system optimizes the heading and winning operations.

similar to those in face openings or in advanced headings, they will be referred to separately for reasons of systemizing and because of other existing problems.

The following illustration shows various conveyor arrangements.



Roadway headings not depending on mining operation and pillar roadways with the ESA system

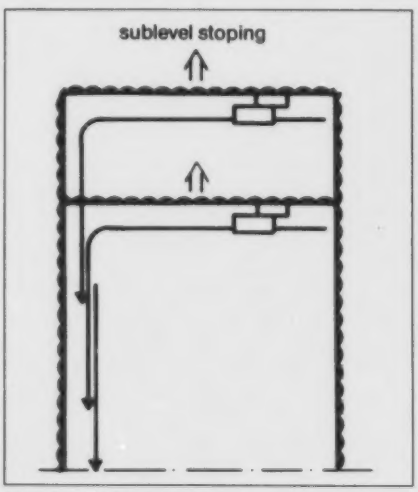


Shortwall face, residual pillar extraction

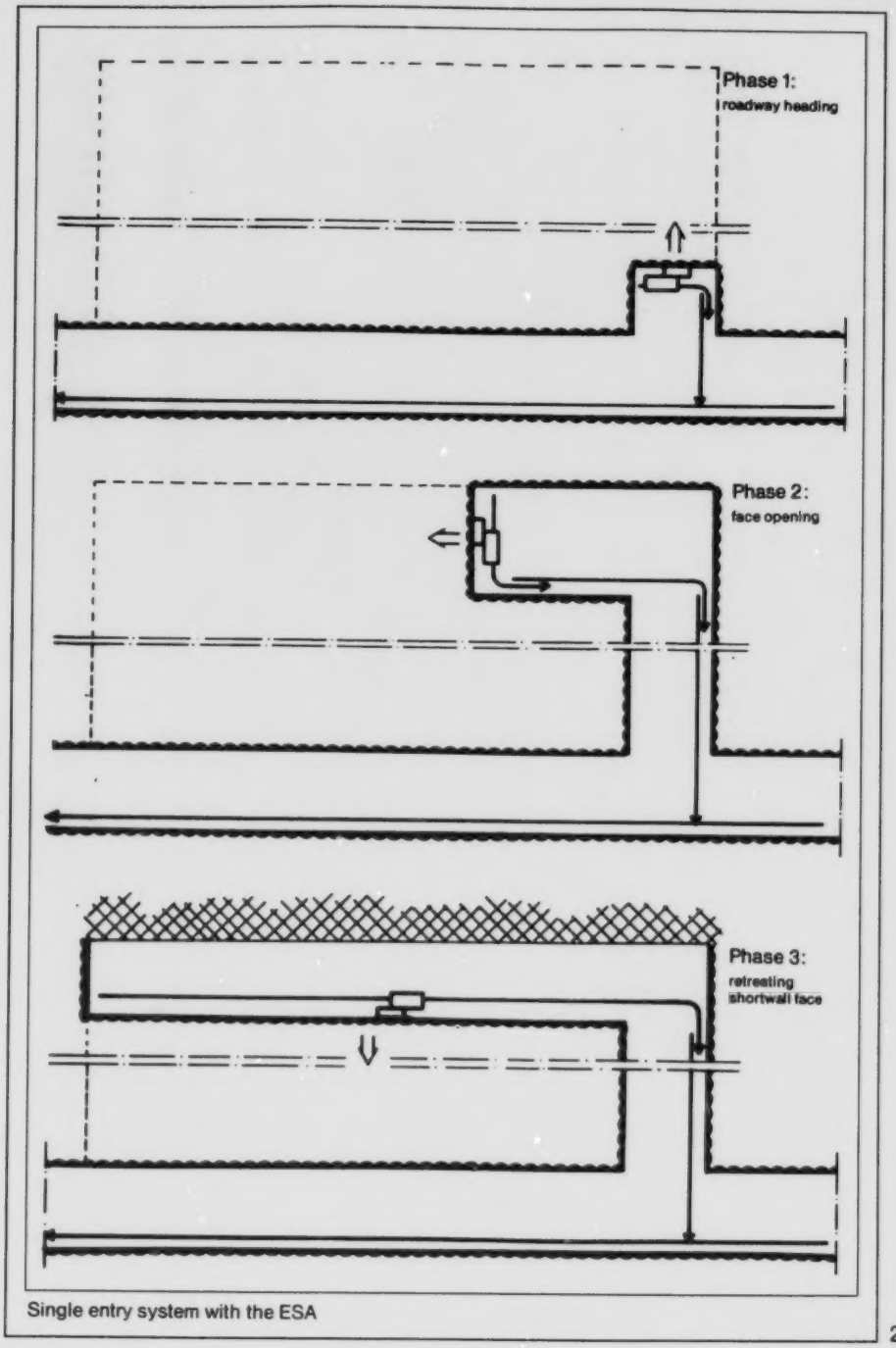
3.3 Mining Methods

The collective term "mining methods" encompasses shortwall workings which serve for regular extraction.

Advancing or retreating workings, which are similar to shortwall faces, with one in-seam roadway are possible as the ESA can swivel the cutting drum right through. Thick seams can be mined in slices by staggered ESA systems in the longwall or shortwall manner. By using the appropriate roof supports thick seams can also be mined by sublevel caving.



Shortwall mining systems



Single entry system with the ESA

ESA-60 machines have fixed drums or can be equipped with 250, 500, 750 mm long ranging arms.

In a single entry mining system the ESA offers the possibility of driving the later haulage road and of opening a face with the same equipment, so that a shortwall face can then be mined on the retreat. Although the necessary approval by the Superior Board of Mines has not yet been obtained in the Federal Republic of Germany, corresponding trials undertaken all over the world will hopefully lead to such an approval as the system not only offers itself for the extraction of residual pillars required for strata control considerations.

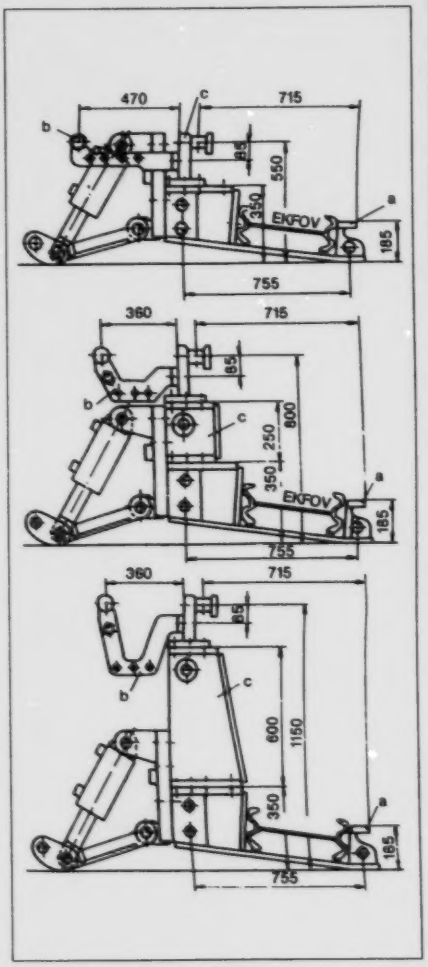
3.4 Adaptation to Various Seam Thicknesses

The possible seam thicknesses are dependent on:

- type of machine
- length of ranging arm
- drum diameter
- face accessories used.

Machines of the ESA-60 series are equipped with fixed drums or, alternatively, with 250, 500 or 750 mm long ranging arms. In order to realize the ESA principle of swivelling the drum along the floor, it is necessary to adapt the accessories to the respective version of the machine.

The accessories consist of the basic body a, the rack bars b, and the intermediate pieces c. The illustration shows the "unit principle of construction" of the ESA-60-L.



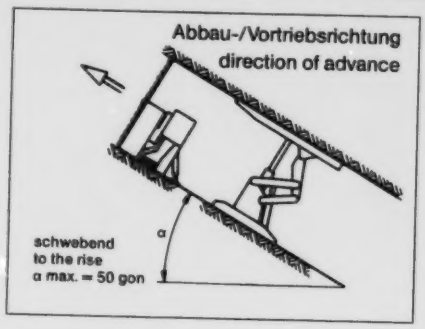
Face accessories (a, b, c)

Maximum cutting heights of the ESA machines: ESA-60-L – 3,20 m, ESA-150-L – 3,85 m, ESA-60-K – 3,62 m, ESA-150-K – 4,80 m.

For wedging the machine into the face a minimum working width of 4,3 m is required. The maximum cutting height which the ESA-60-L can reach amounts to 3,2 m. The lower limit of the cutting range lies at present at 1,4 m.

Due to its length the machine type ESA-150-L is at present only being used on shortwall faces. The machine can be equipped with ranging arms of 1050, 1335 and 1600 mm in length, and in conjunction with the appropriate cutting drum a range of 1,80 m to 3,85 m can be cut.

The series ESA-60-K and ESA-150-K have an even greater range of movement as their machine body can be lifted. This allows for reaching maximum cutting heights of 3,62 m (60-K) and 4,8 m (150-K).

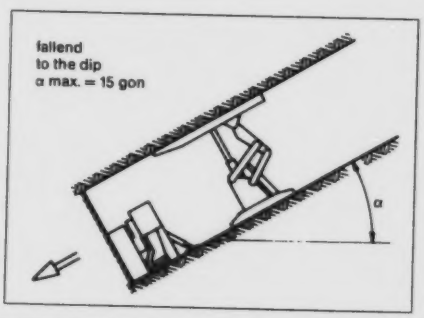


3.5 Operation in Inclined Seams

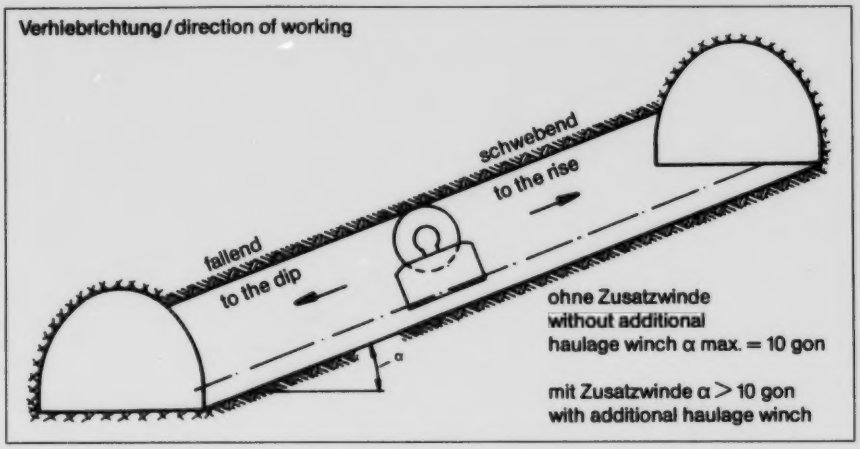
Up to now ESA shearers have been installed on gradients up to 50 gon to the rise in the direction of advance and up to approx. 15 gon to the dip.

With regard to inclinations in the direction of advance the following must be observed:

According to § 103 of BVOSt of the Superior Board of Mines of Northrhine-Westphalia machines for winning coal by cutting must be secured against sliding down, independent of their haulage system, when operating on a gradient exceeding 10 gon and when these machines are conveyor mounted.



In inclined seams the ESA machines are secured against sliding down either by two haulage units, by one haulage unit and an additional brake, or by mechanical means.



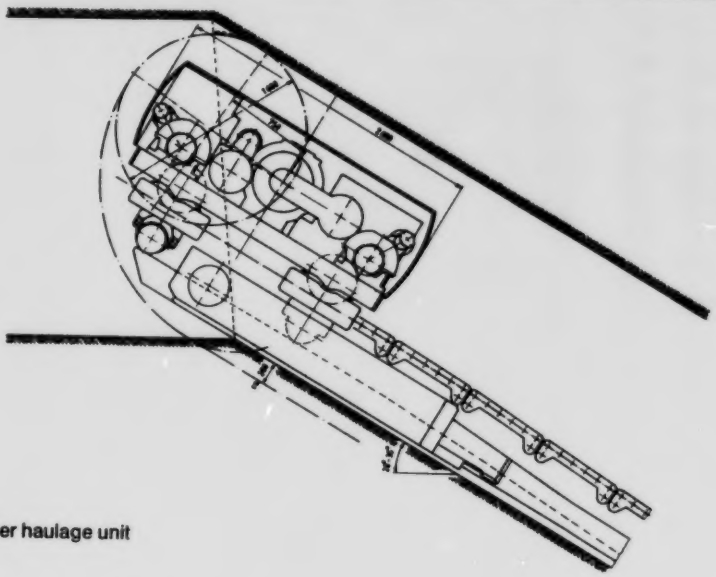
There are at present 3 possibilities for ESA machines to comply with this regulation:

1. **Operation with two haulage units,** each of them being equipped with a hydraulically lifted lamellar brake. In the case of the failure of one brake or a defective gearing, braking is ensured by a hydraulical safety arrangement.

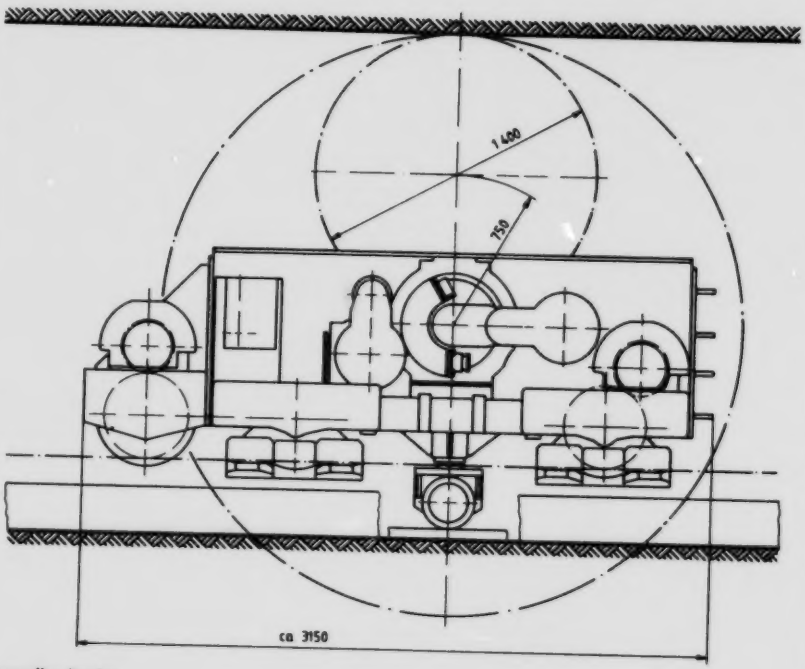
   In comparison to a machine with one haulage unit the haulage pull doubles at constant travelling speed.

2. **Operation with one haulage unit and additional brake.** Positive connection of the additional brake is ensured by a second rack wheel with flanged-on, hydraulically lifted lamellar brake.

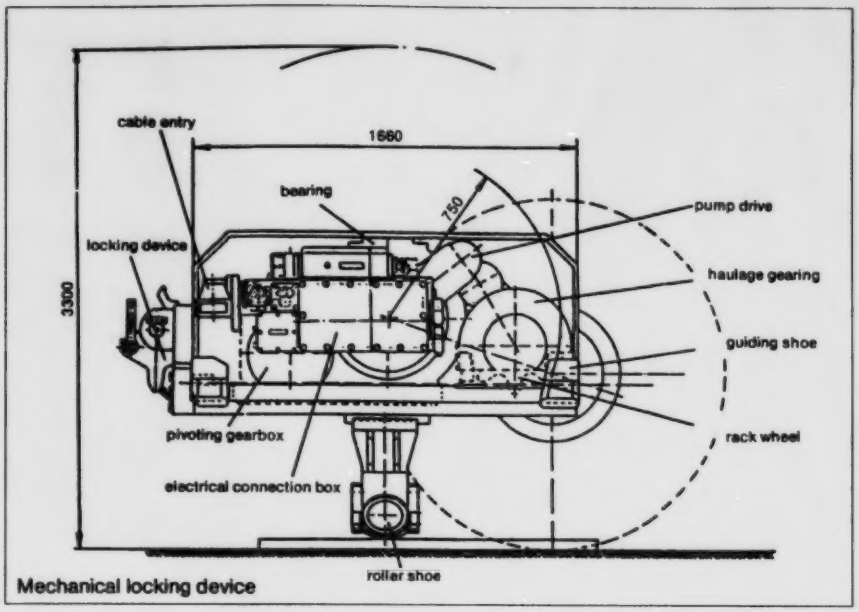
3. **Mechanical safety means.** This consists of a safety catch which slides over the rack pegs during the upward traverse. Should the ESA slide down due to the breakage of a tooth or a gearing defect of the haulage unit the safety catch is engaged in the rack bar and must be manually released for the downward traverse. The coalface itself then prevents the machine from sliding down. The system has been approved by LOBA under a letter of no objection.



Booster haulage unit



Lamellar brake



Mechanical locking device

4. System Units for the Various Operating Ranges

4.1 Roof Supports in Wide Face Openings

The width of a face opening depends on the following criteria:

- type of face equipment to be installed
- seam thickness
- roof properties
- mining method
- time at which the shield supports are installed.

This also determines the type of the roof supports to be used.

The roof in the area behind the working face must be supported as soon as possible. If linear or triangular roof supports are installed, the roof is supported by rails or Van-Wersch roof bars. Sliding bars can also be used for forepoling. The final roof supports can be set immediately behind the ESA accessories. If single props are set it must be ensured that the distance from the face to the first prop is at least 1595 mm because of the width of the machine.

Chock shield supports offer a high supporting resistance near the working face due to their sliding front cantilevers. A further advantage is the integration of the roof supporting, abutment and shifting functions within one system.

The final roof supports which are usually of the frame type are set behind the shields. However, the installation of the pilot shields necessitates precise coordination of the entire system and may entail problems due to the trample effect in the case of a bad floor.

The roof supports in a face opening can be represented schematically according to the type of forepoling.

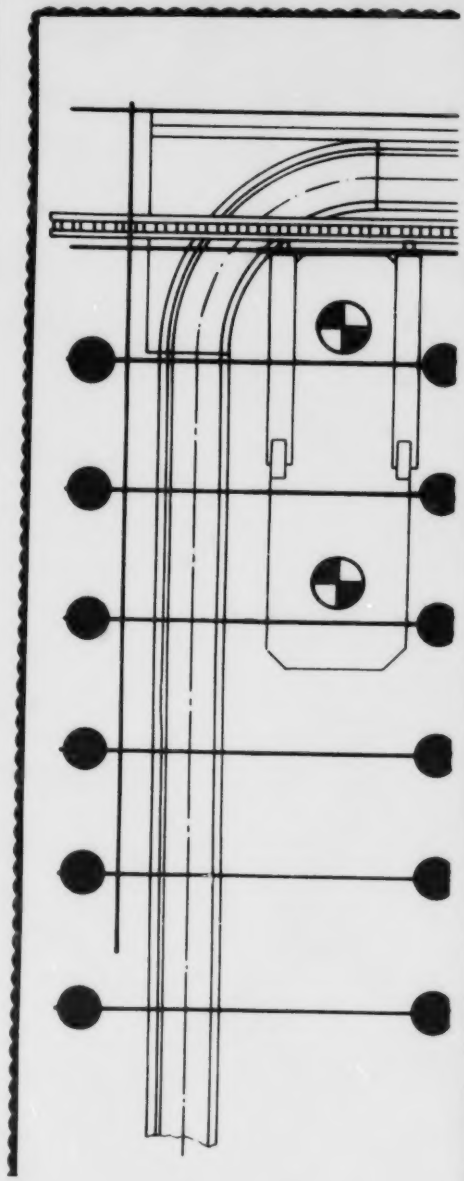
Single Props (at the working face)

- triangular supporting system with Van-Wersch double wedge roof bars and hydraulic props;
- frame type supporting system with sliding rails, forepoling rails, and hydraulic props;
- frame type supporting system with forepoling by sliding roof bars.

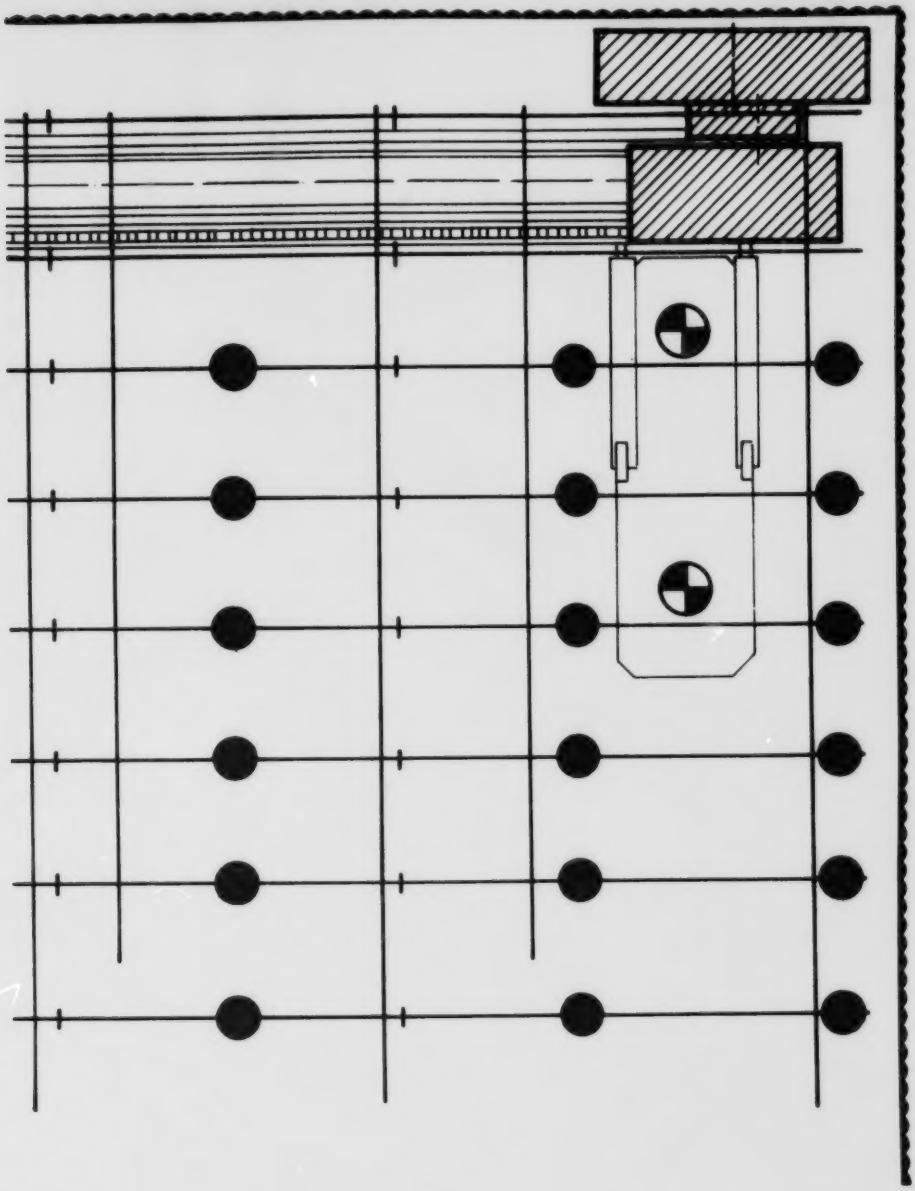
Shield/Chock Shield Roof Supports (at the working face)

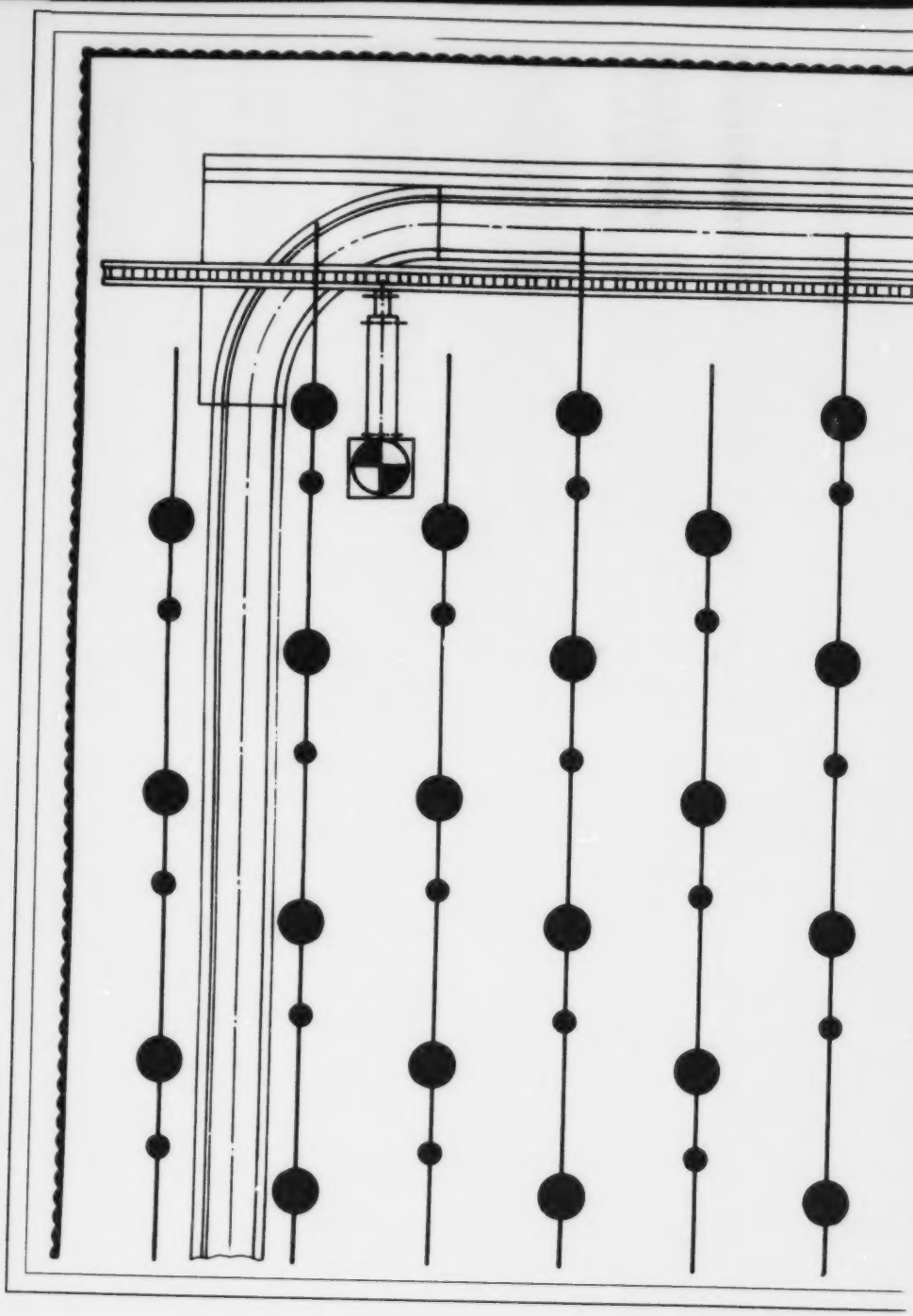
- frame type supporting system with subsequent roof bar forepoling and frame type roof supports guided in parallel;
- subsequent roof bolting.

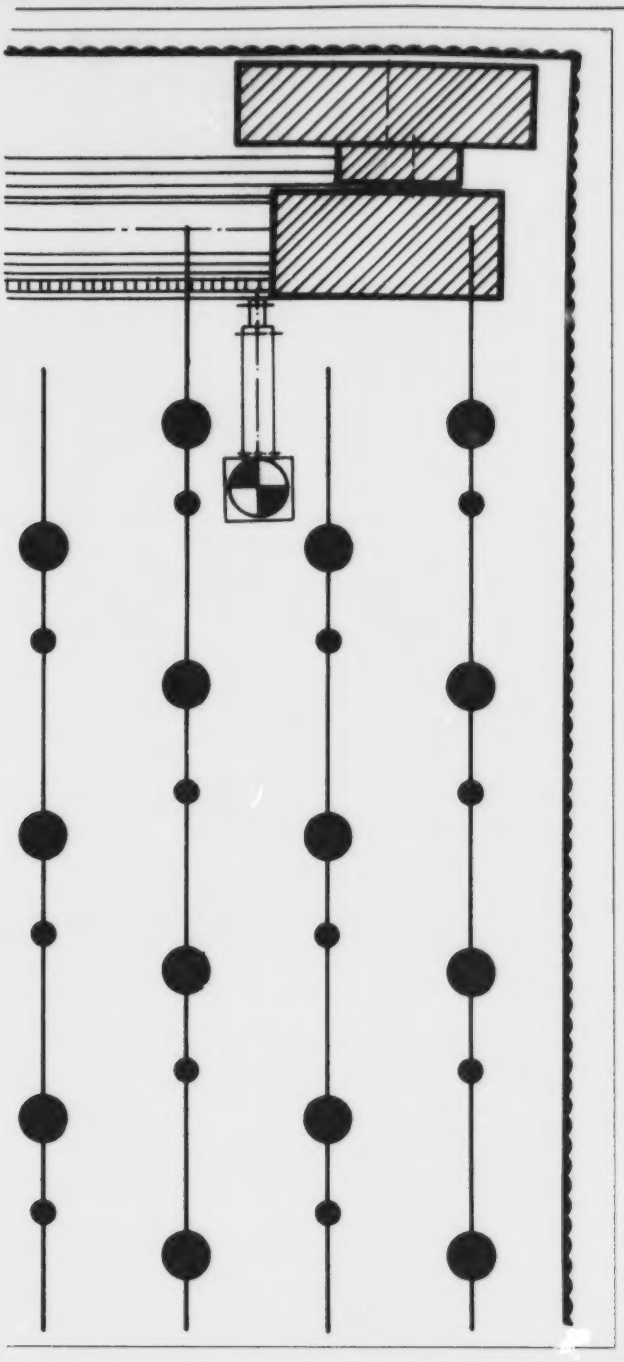
The following illustrations show some practical examples:



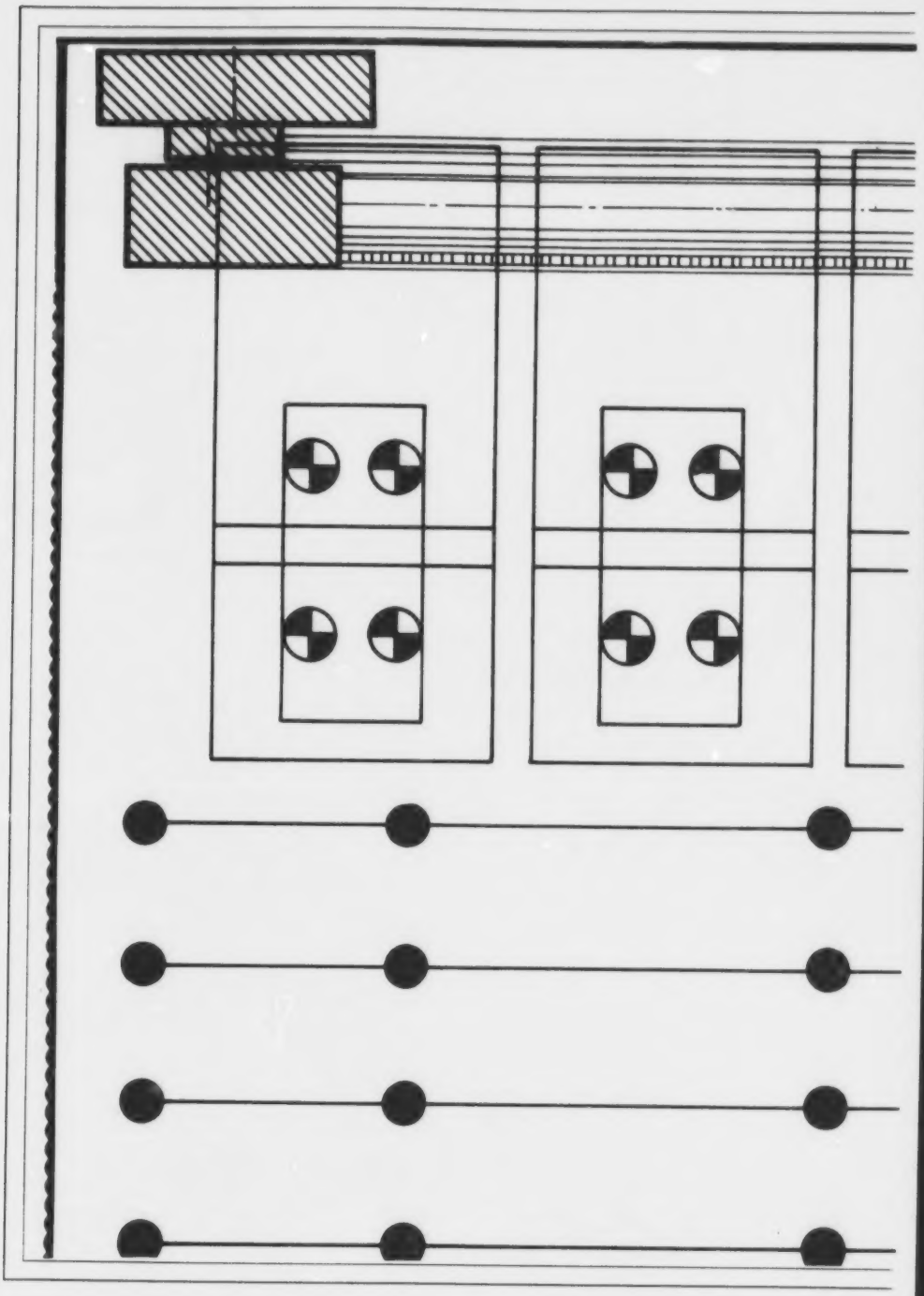
Frame-type roof supports
consisting of rails, cantilever rails and hydraulic props

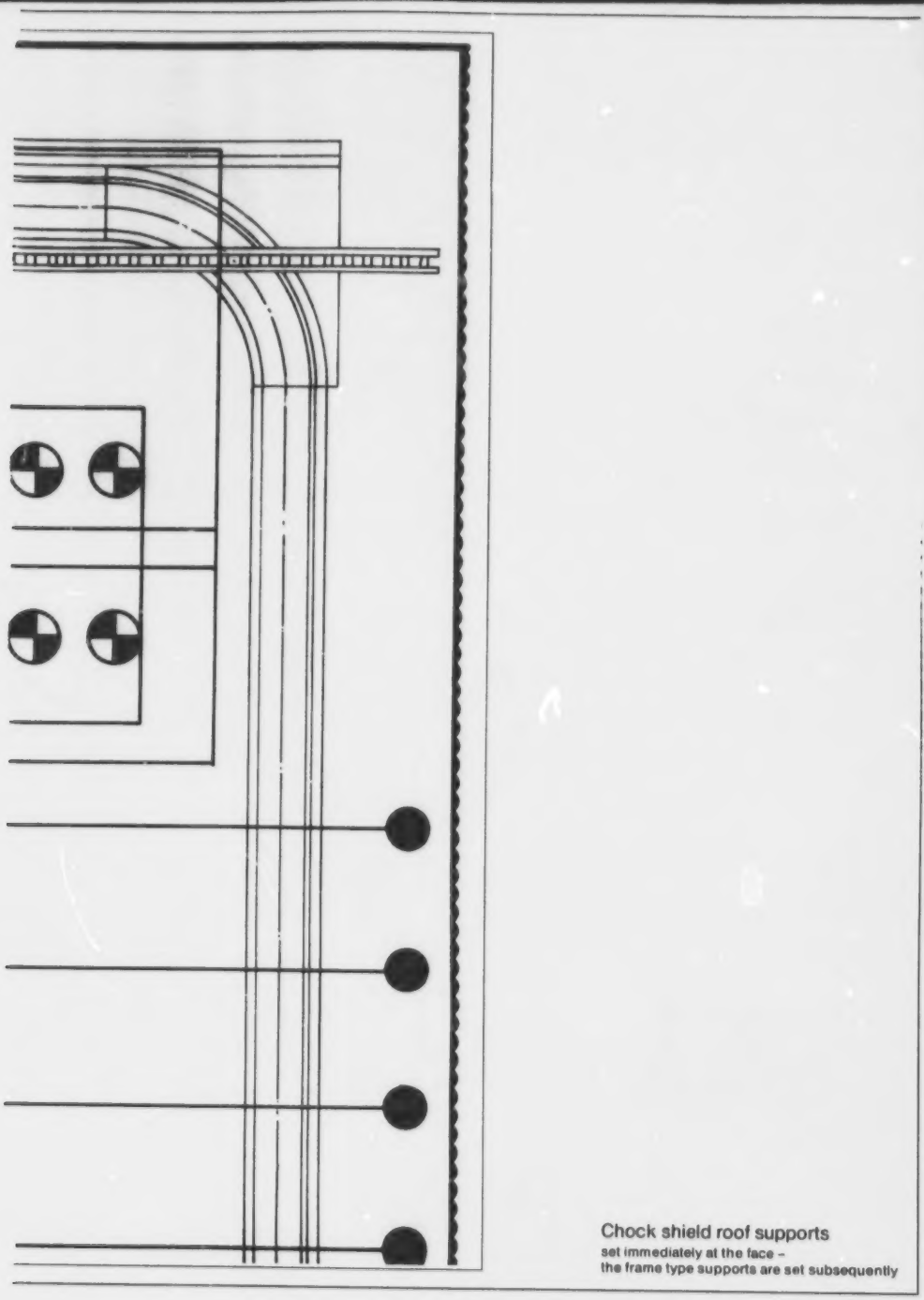




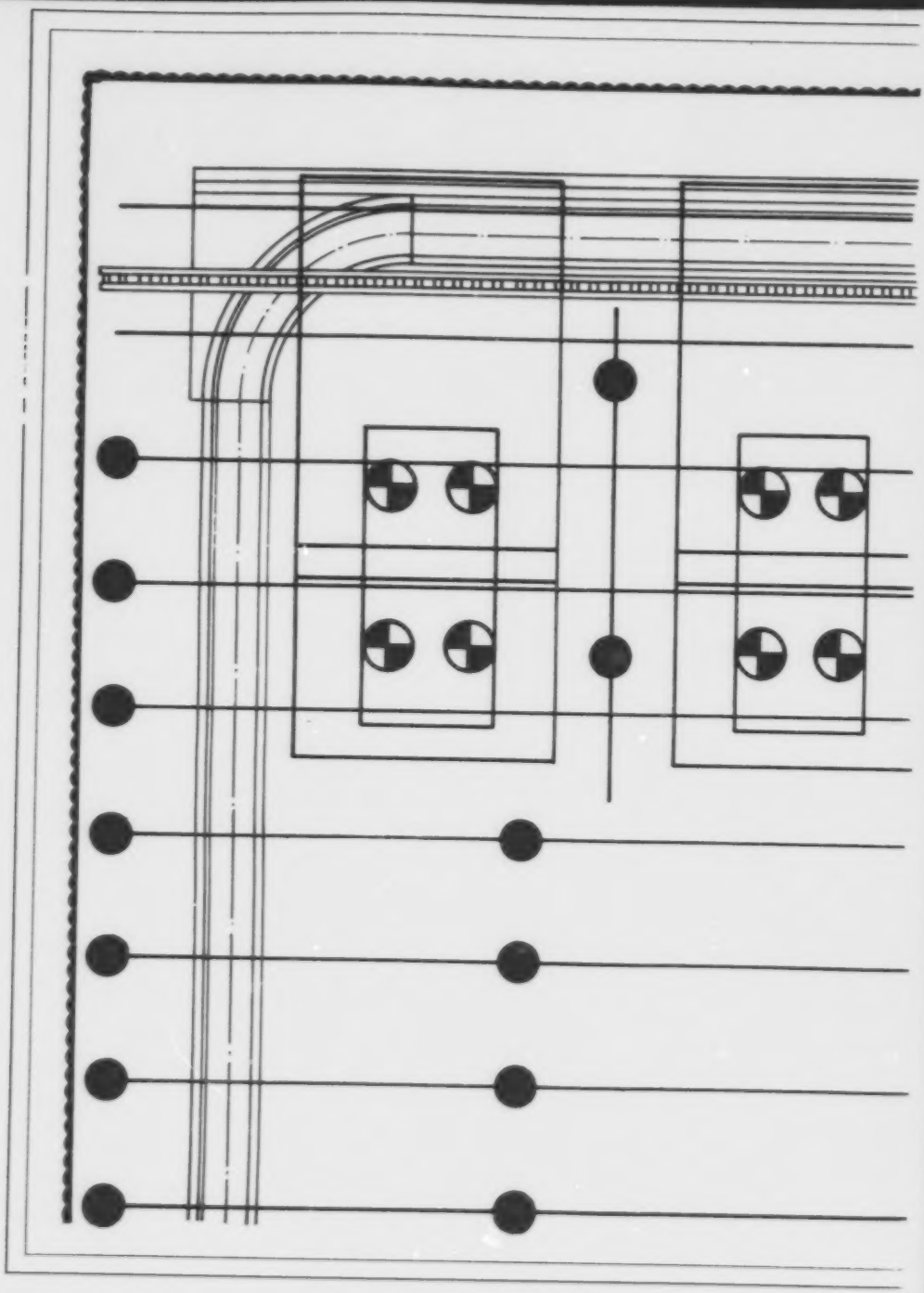


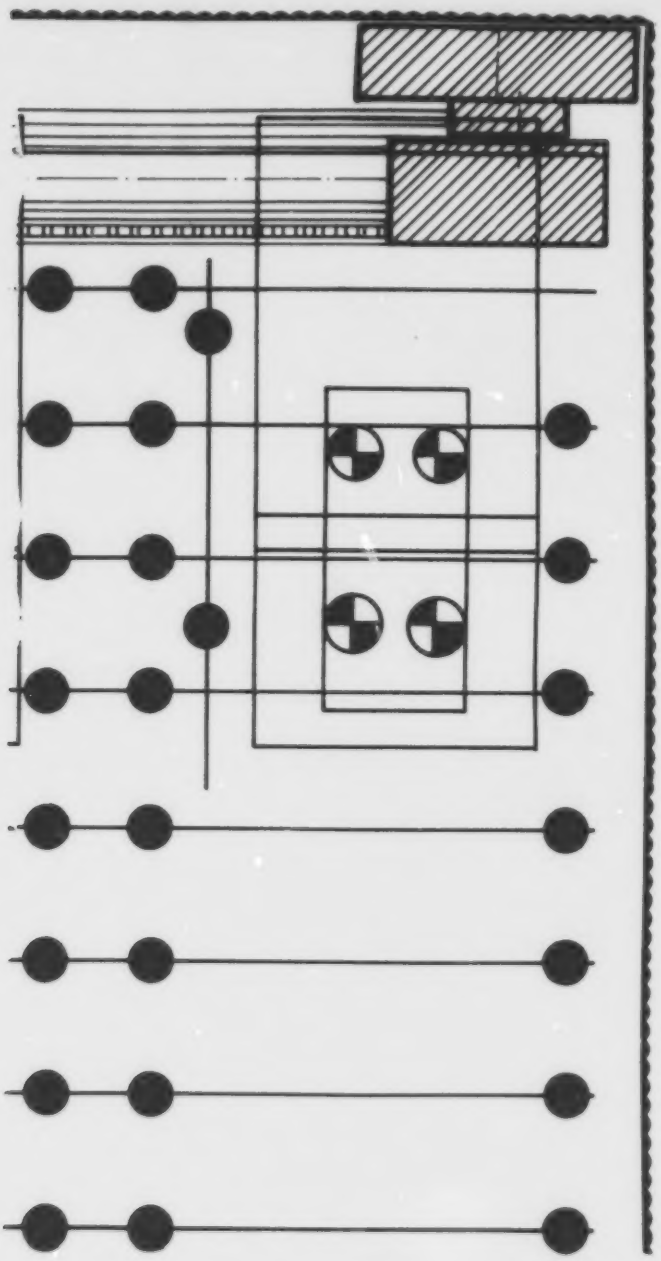
Triangular roof supports
consisting of van-Wersch roof bars and hydraulic props



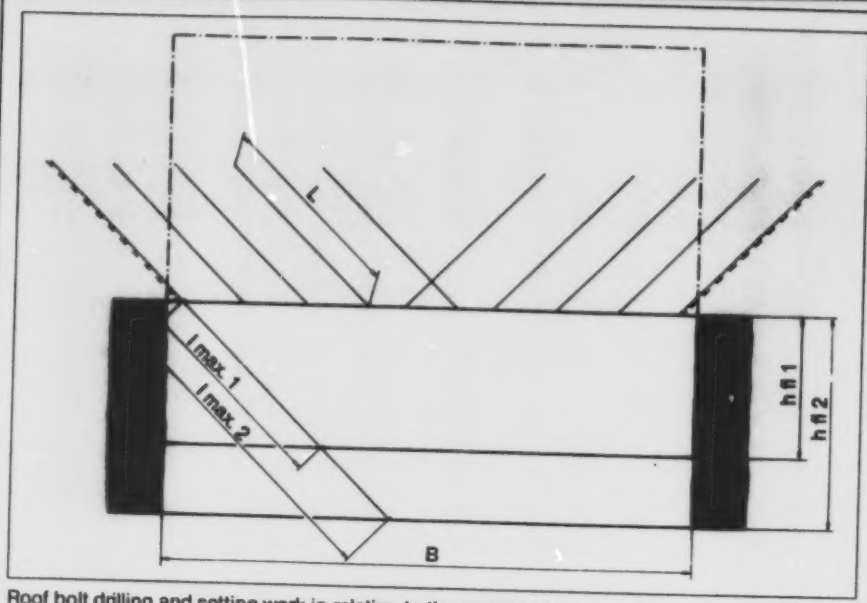


Chock shield roof supports
set immediately at the face –
the frame type supports are set subsequently





Chock shield roof supports
set immediately at the face – the frame type supports with cantilever roof bars are set subsequently



Roof bolt drilling and setting work in relation to the seam thickness

width of face opening	B	= 6,50 m	spacing of roof bolts in the direction of advance	r	= 0,82 m
seam thickness	h_{FL1}	= 1,60 m	max. available drilling length	$l_{max.\,1}$	= 2,25 m
	h_{FL2}	= 2,40 m		$l_{max.\,2}$	= 3,35 m
length of roof bolts	L	= 2,20 m			

The following experience has been gained with the setting of roof bolts behind the ESA system:

So as to limit the amount of time involved in setting the roof supports drilling should be carried out "over the full length" without extension rods, if possible. For geometrical reasons the application is confined by the seam thickness. There must be sufficient space for the drill rod and the drill carriage or the drill hammer in the case of rotary percussion drilling.

The problems involved are shown for two seam thicknesses as an example.

Under the given requirements drilling is not possible in a seam thickness of 1,60 m on account of the max. length of the drill of 0,5 m.

The roof bolt drilling and setting work must fulfil a further requirement if the roof bolt line distance exceeds the cutting depth of the drum. In this case the mechanical equipment for roof bolting must be separated from the remaining heading equipment as cutting of a full round takes less time than roof support setting. If the roof conditions permit roof bolting and hence the roof support work should be separated from the heading system.

4.2 Conveyors in Face Openings and Possible Conveyor Overlapping

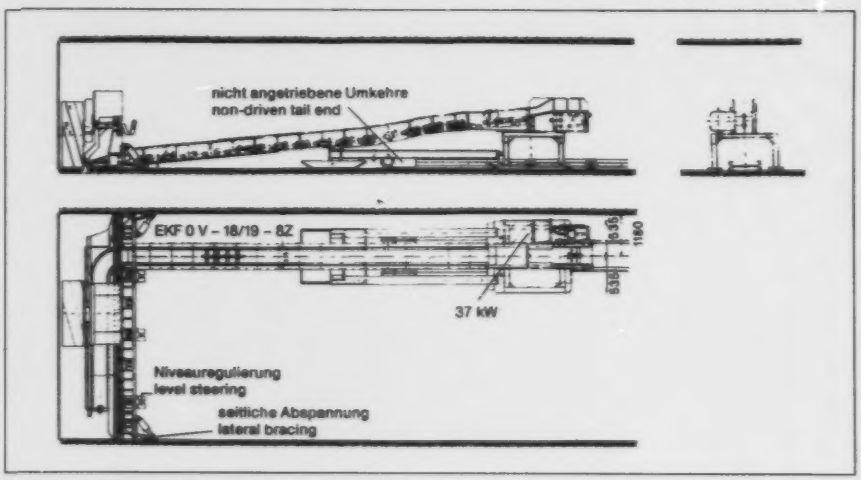
An EKF-O chain conveyor of the 'L' version is the standard equipment for the evacuation of the material cut and loaded. The position of the evacuating section depends on the later handing of the face, i.e. on the position of the face conveyor which is successively installed during the course of the face opening.

Other arrangements of the heading conveyor in face openings with single props as, for instance, those with the evacuating section passing through a further curve are not recommendable not only on account of the higher frictional forces due to additional angular deflections, but mainly also due to the lack of coordination of the roof supporting work, as the single props already set during shifting must be removed and then reset again.

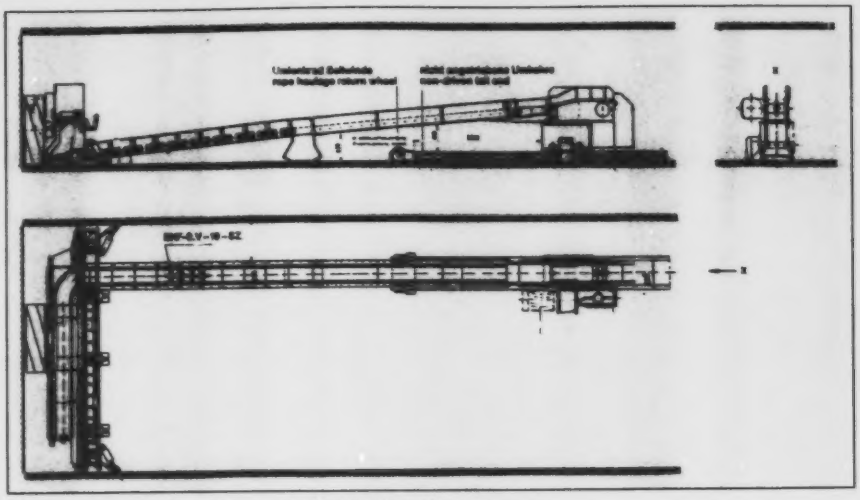
The question regarding the equipment of the curved conveyor with a rigid or a roller curve can only be answered if the width and the height of the face opening are known, because roller curves are generally larger and therefore obstruct the view of the mining area or delay the setting of the roof supports.

The heading conveyor overlaps the final face conveyor generally by the length of the daily advance, so that the face conveyor can be extended during the night shift.

The face conveyor is extended immedia-



Conveyor overlapping with support on the floor

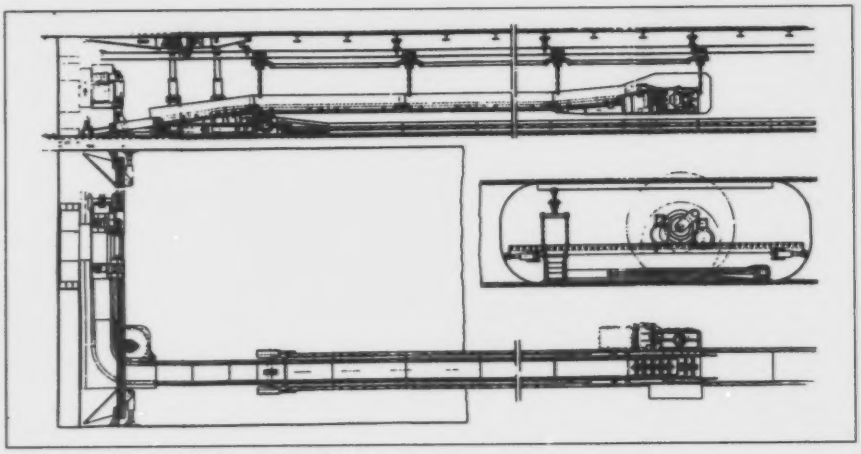


Conveyor overlapping with support on the final face conveyor

tely at the entry of the face opening by the installation of additional line pans.

The face opening projects realized so far used two distinct arrangements of the heading conveyor:

- The heading conveyor is supported either on the floor or on the final face conveyor.
- In thick seams the overlapping section of the heading conveyor is suspended from a monorail, an arrangement which is particularly advantageous on a weak floor.



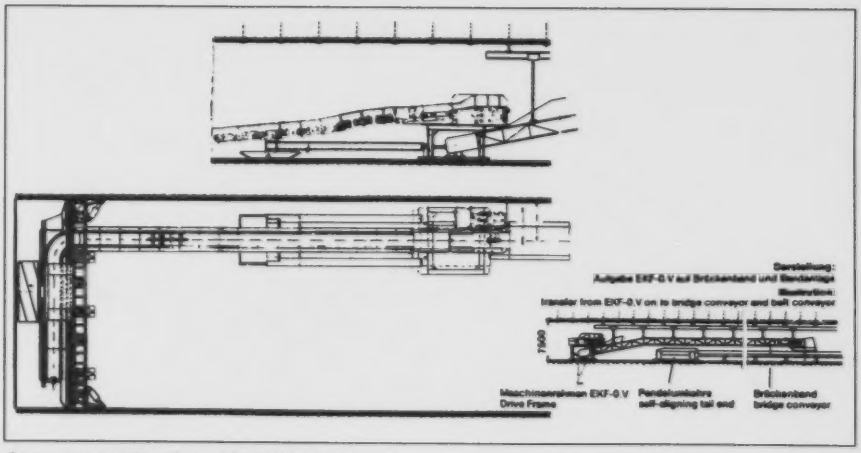
Conveyor overlapping in conjunction with a monorail

4.3 Conveying Means in Roadway Headings and Possible Overlappings

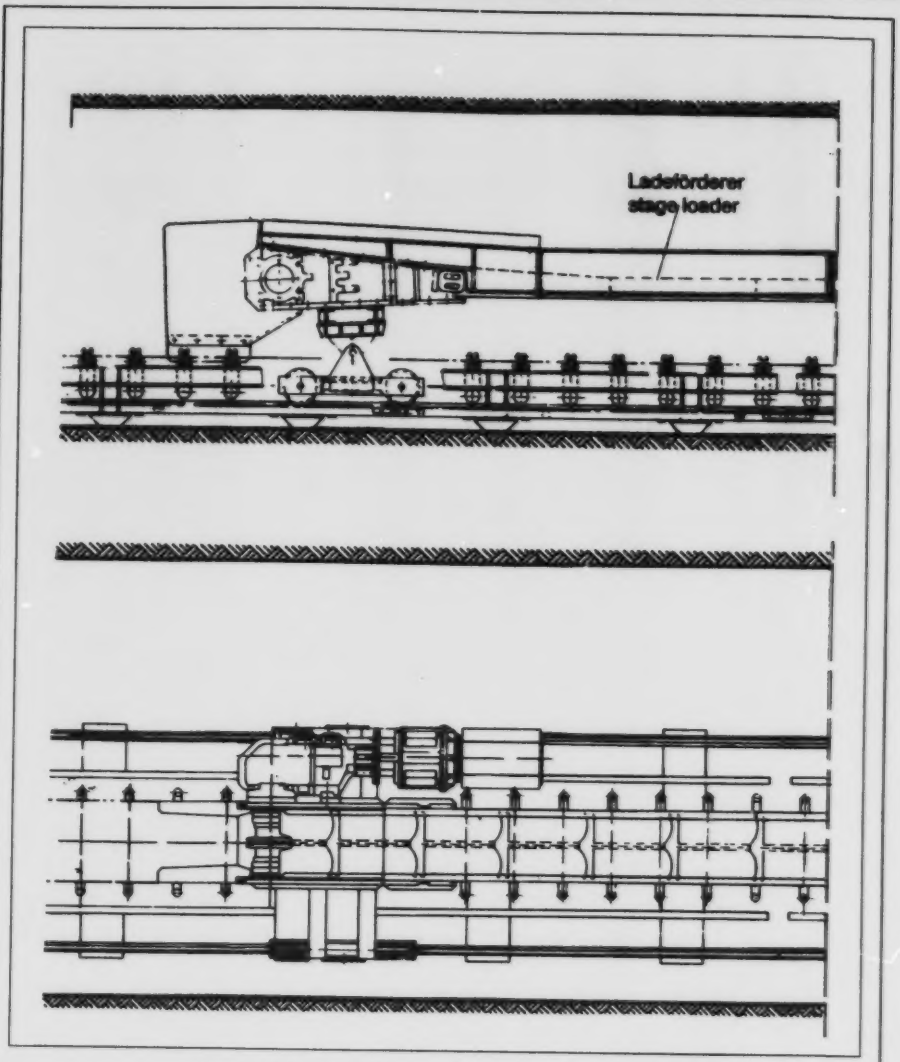
The conveying means in roadway headings and pillar roads can be distinguished as continuous or shuttle conveyors, and the continuous conveying means can be distinguished further according to type and overlapping arrangement. The continuous conveying means comprise belt and scraper chain conveyors, whereas the shuttle conveyors are actually shuttle cars particularly in pillar workings.

In roadway headings which are independent of the mining operations the conveying means are comparable with those in face openings, but in general belt conveyors are being installed due to the larger overall length. The arrangement of the conveying means in advanced roadway headings still presents problems, particularly in cases where the material must be evacuated through the face and where an additional stage loader is required for dinting work.

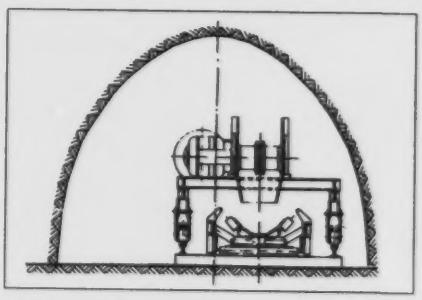
The following illustrations show the conveying means installed up to now and particularly their overlapping arrangements.

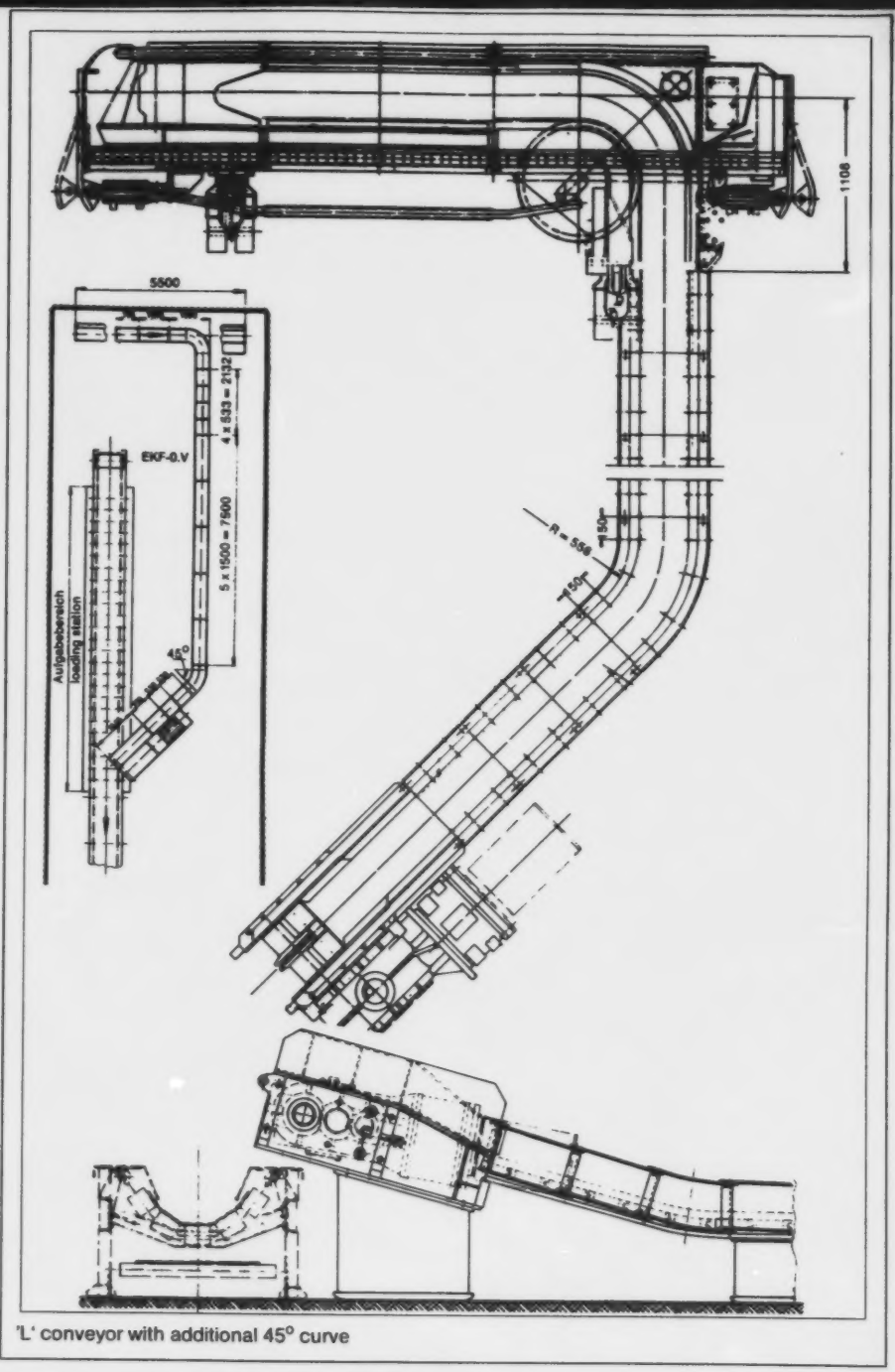


Conveyor overlapping with bridge conveyor

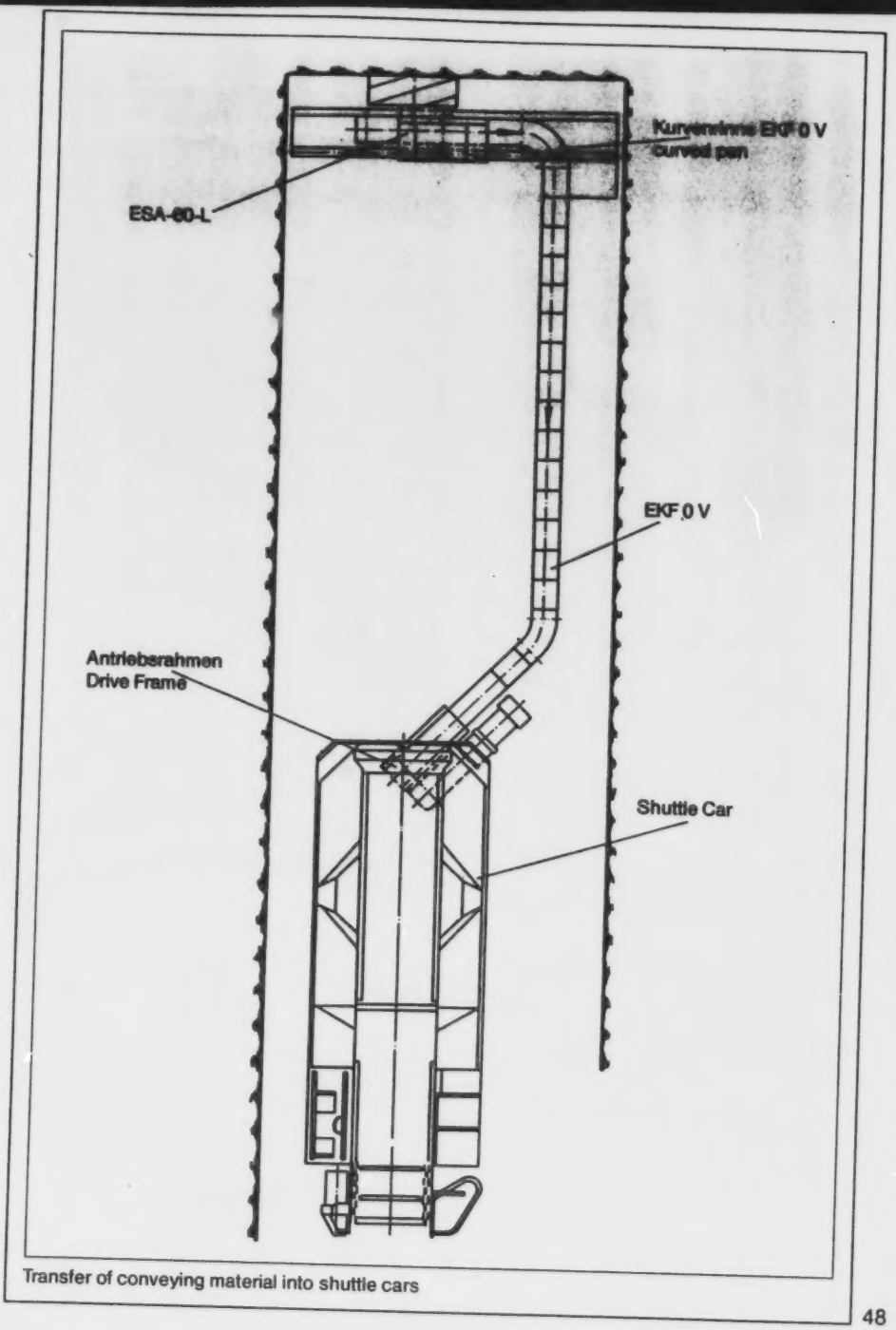


Support with roller shoes

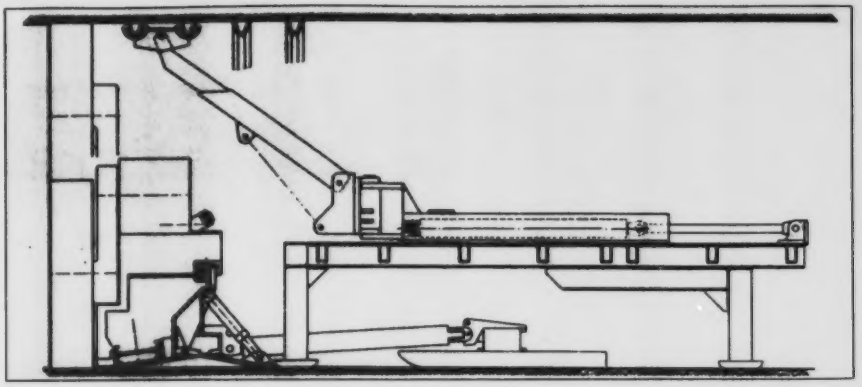




'L' conveyor with additional 45° curve



Transfer of conveying material into shuttle cars



4.4 Roof Support Setting Aids in Roadway Headings

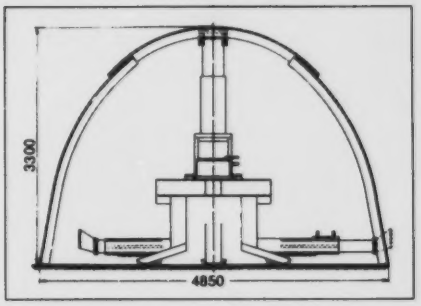
The main requirements to be met by roof support setting aids are:

- facilitation and acceleration of the roof support setting work;
- forepoling of the assembled roof bars and simultaneous provision of high supporting forces for the roof until the face props are set.

Some possibilities have already been tried in conjunction with the ESA system and they are considered as being feasible, but the requirement regarding the introduction of the roof support resistance has not been met satisfactorily yet.

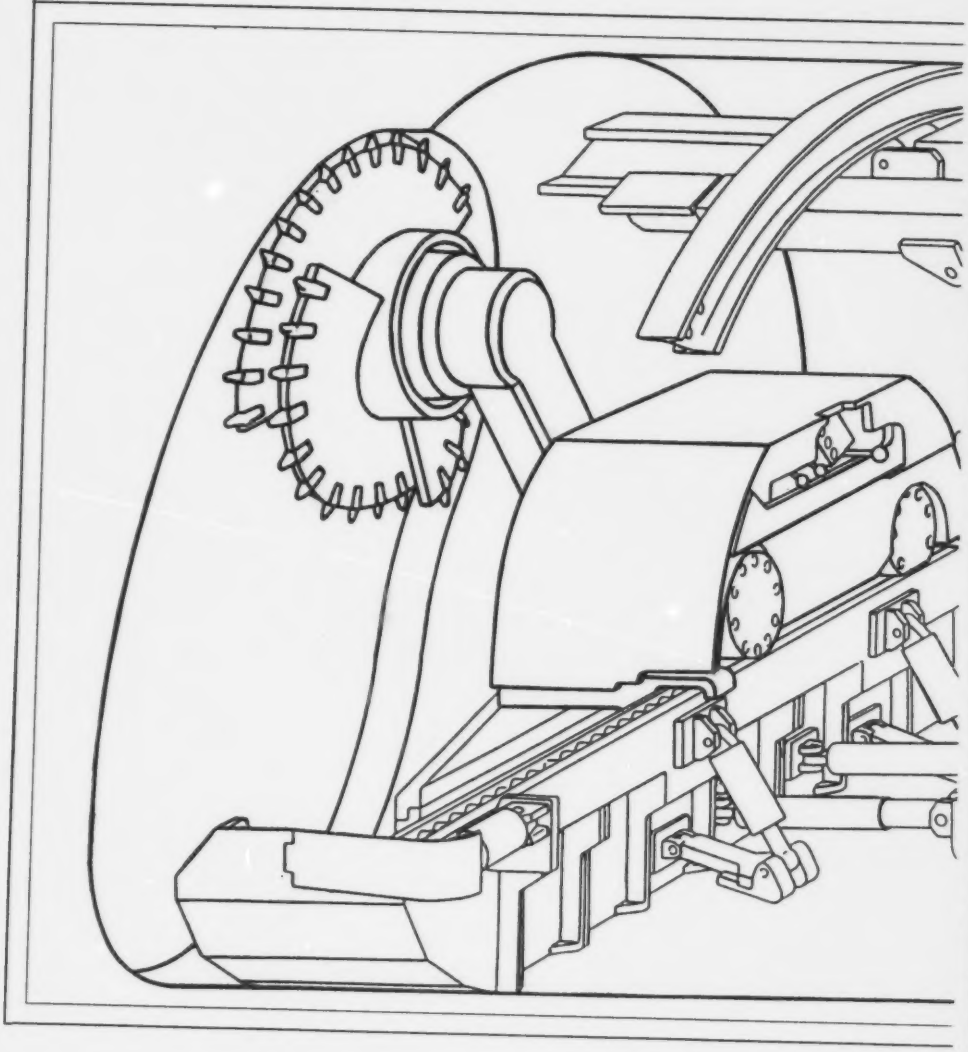
The following roof support setting aids should be mentioned:

- Figure a shows a roof support setting aid with a three-part roof section. The floor support and the roof canopy are split. The system is of the self-advancing type and serves simultaneously as a shifting device.
- The arch setting device is mounted on the floor and is retained in position by lateral bracing cylinders. The setting device is arranged on a trailed platform behind which the necessary space required for storing and picking up the pre-assembled roof sections is provided. By retracting and pulling back the substructure, the setting device can accommodate two line pans simultaneously (fig. b).
- Another possibility with a similar type of line pan accommodation is shown on figure c.

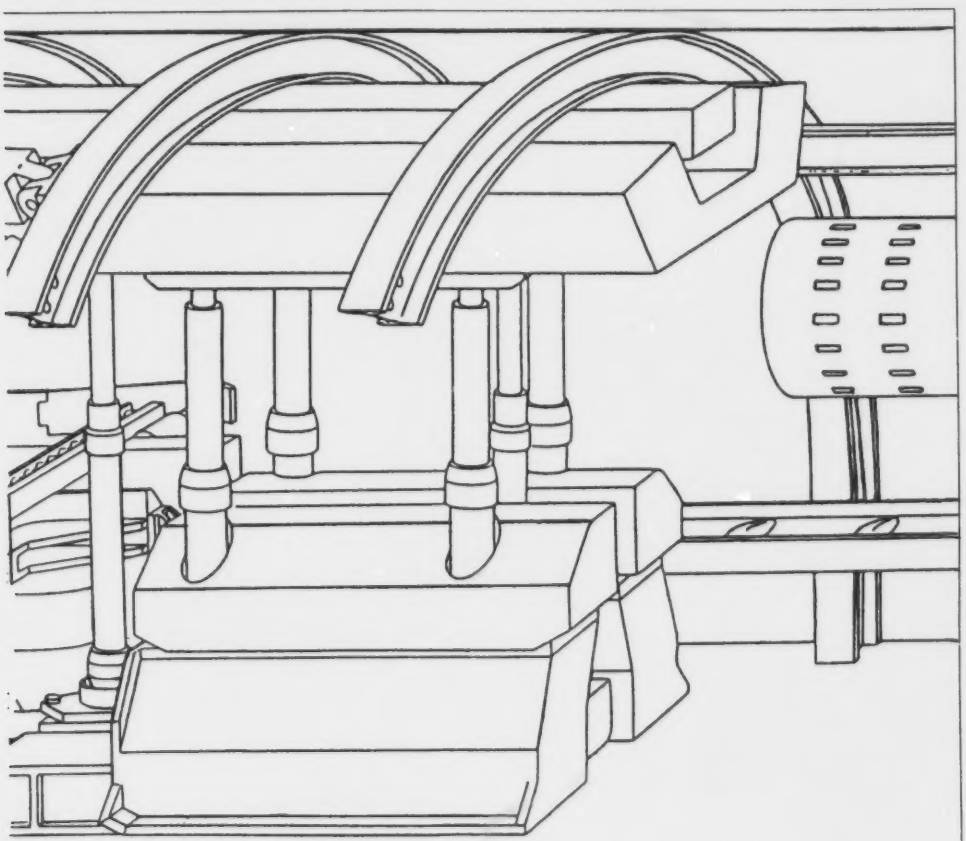


b) Roof support setting aid with lateral bracing cylinders

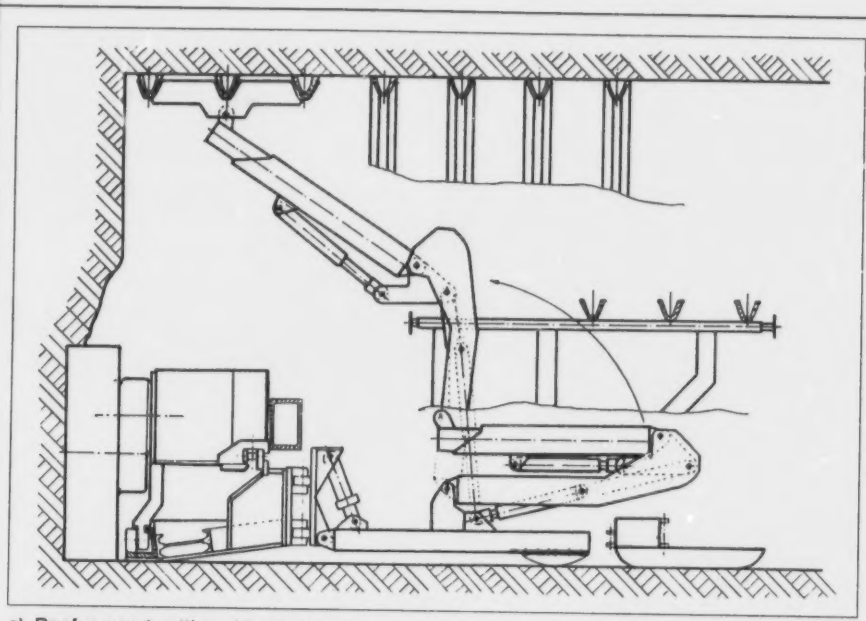
The setting aid for the arch suppo
self-advancing type and serves si
The floor support and the roof ca



ts with 3-part roof section is of the ultaneously as a shifting device. opy are split.



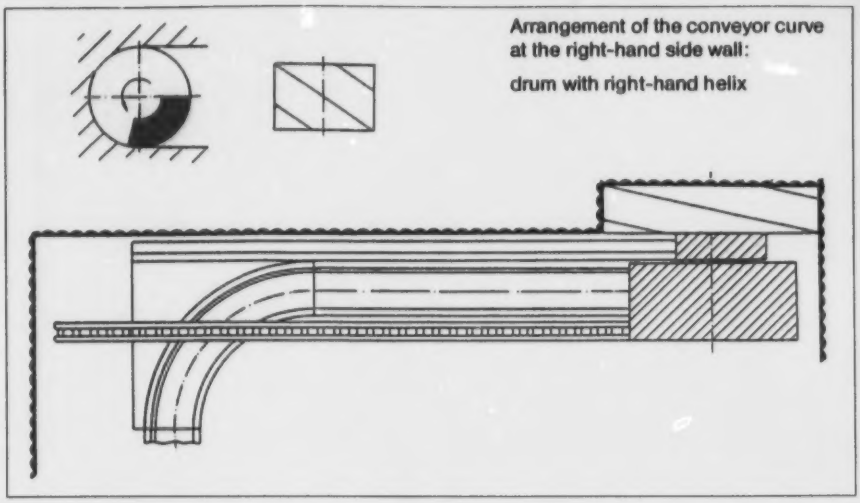
a) Roof support setting aid with split floor support and roof canopy



c) Roof support setting aid with lateral bracing cylinders of more recent design

4.5 Design of the Cutting Drum in Relation to the Arrangement of the Conveyor Curve

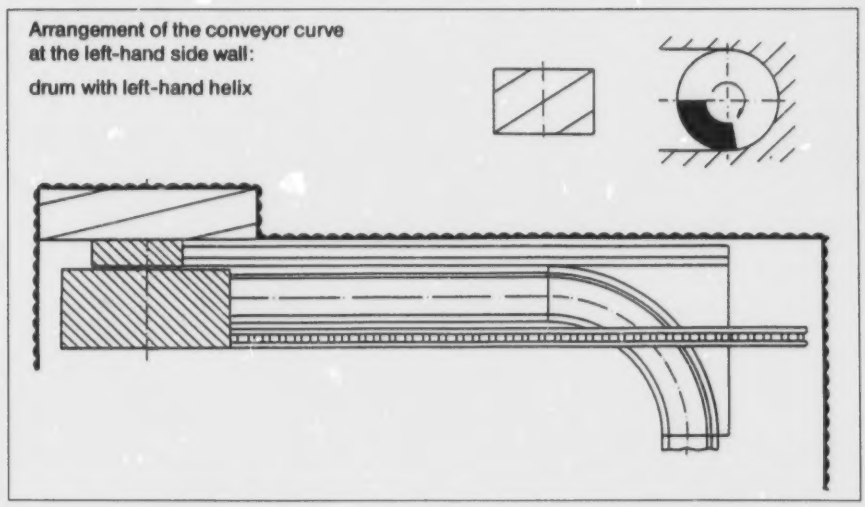
Conveyors equipped with a curve are generally used in face openings and roadway headings for evacuating the material cut and loaded. The arrangement of this curve on one side of the face increases the loading distance from the cutting drum to the conveyor in this area. In order to counteract inefficient loading the cutting drum can be adapted to the arrangement of the curve. A cutting drum having a right-hand helix has its maximum loading capacity in quadrant III on the goaf side due to the drum geometry. A cutting drum with a left-hand helix loads the mineral cut in quadrant IV.



Arrangement of the conveyor curve at the right-hand side wall:

drum with right-hand helix

As the cutting drum must have its maximum loading capacity on the side facing the conveyor in order to optimize the loading efficiency in the area of the curve, the following is recommended: a drum with a left-hand helix for a curve on the left-hand side of the face and a drum with a right-hand helix for a curve on the right-hand side of the face.



Arrangement of the conveyor curve at the left-hand side wall:

drum with left-hand helix

ESA machines can be equipped with a cutting drum or a sumping-in drum.

5. Special Methods

5.1 Operating Cycles

ESA machines can be equipped with a cutting drum or a sumping-in drum. The following illustrations show the cutting-in and sumping-in processes as well as further methods in different seam thicknesses and openings:

Operational sequence with ESA-60-L in a face opening (seam thickness = drum diameter):

Fig. 1: The ESA-60-L wedges itself into the face as the conveyor is advanced on one side.

Fig. 2: The first half of the full section has been cut.

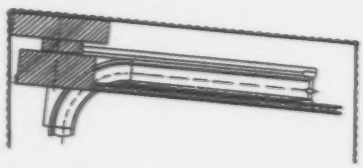


Fig. 3: The conveyor is fully advanced, so that the ESA-60-L can cut the remaining wedge.

Fig. 4: After the extraction of the remaining wedge the roof supports can be advanced.

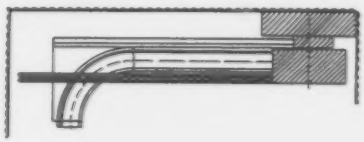
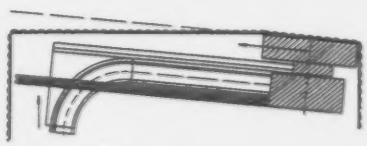


Fig. 5: The initial position is re-established by advancing the conveyor on one side. In order to avoid any transverse loads on the drum, it is kept rotating during shifting.



Operational sequence with ESA-60-L in a face opening (seam thickness exceeding the drum diameter):

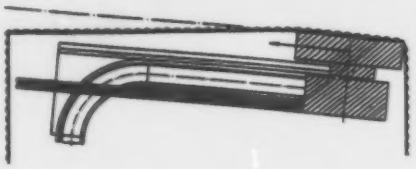
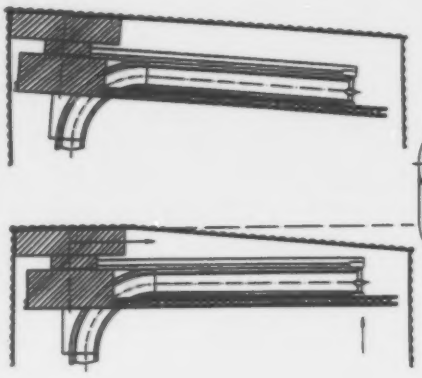


Fig. 1: The ESA-60-L wedges itself into the roof coal and the conveyor is advanced on one side.



Figs. 2a, 2b: The first half of the full section has been cut at the roof. The conveyor is fully advanced, so that the ESA-60-L can start to cut the remaining wedge in the roof coal.

Fig. 3: The cutting drum cuts from the roof coal to the floor coal. The cantilever or sliding roof bars of the roof supports can then be set. The ESA-60-L starts to cut the floor coal over the full length of the working face.

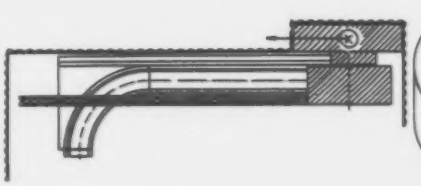
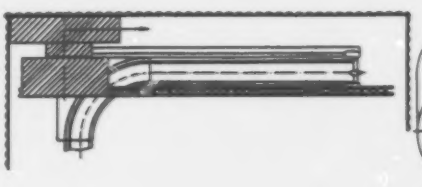
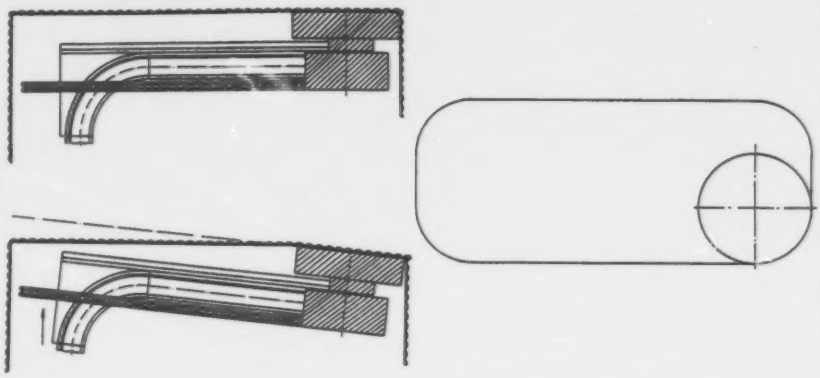


Fig. 4: A cleaning up run is performed.



Figs. 5a, 5b: The full section is cut. The initial position is re-established by advancing the conveyor on one side. In order to avoid any transverse loads on the drum, it is kept rotating during shifting.



Operational sequence with ESA-60-L in a face opening (seam thickness greater than drum diameter) – "spiralling in":

Fig. 1: The conveyor is advanced on one side by half the depth of the drum. The drum wedges itself into the roof coal.

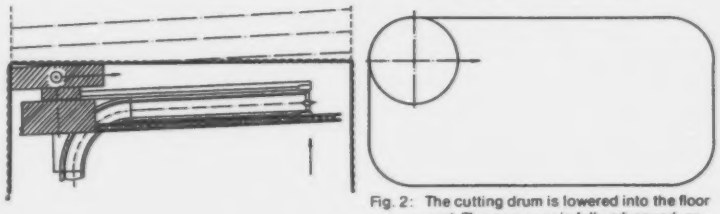
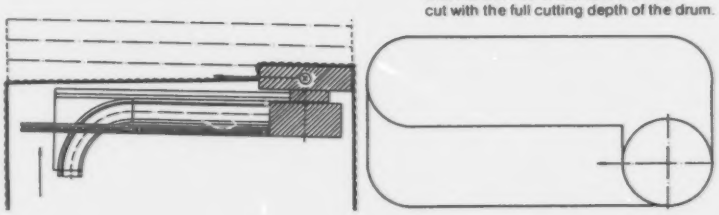


Fig. 2: The cutting drum is lowered into the floor coal. The conveyor is fully advanced, so that the ESA-60-L can take the floor cut with the full cutting depth of the drum.



After the floor cut the cutting drum is raised into the roof coal. At the same time the conveyor must be advanced on one side. Cutting takes place in a sprial manner with the full depth of the drum. The roof cut is taken from the left to the right and the floor cut is taken from the right to the left. The working face is always at an angle to the direction of advance.

Operational sequence with ESA-60-L in a face opening (seam thickness greater than diameter of sumping-in drum):

Fig. 1: The axial advance for sumping in is initiated by the shifting cylinders of the conveyor. The sumping-in drum cuts into the roof coal.

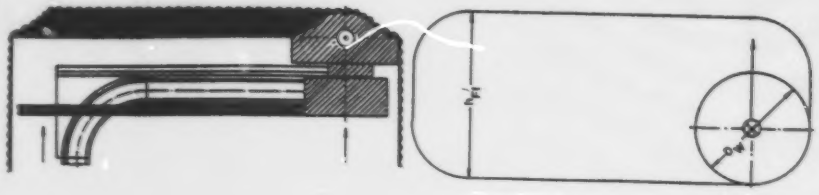


Fig. 2: The ESA-60-L starts to cut the roof coal over the full length of the working face.

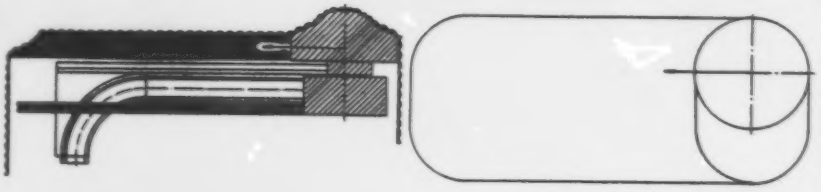


Fig. 3: The sumping-in drum cuts from the roof coal into the floor coal. The cantilever or sliding roof bars of the roof supports can then be set.

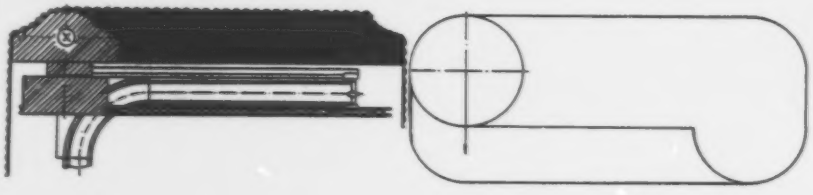
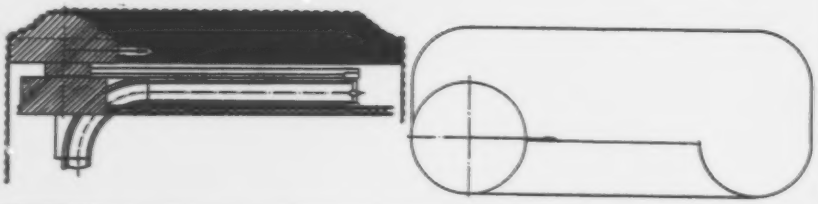


Fig. 4: The ESA-60-L starts to cut the floor coal over the full length of the working face.



Operational sequence with ESA-60-L in a face opening (seam thickness = diameter of sumping-in drum):

Fig. 1: The axial advance for sumping in is initiated by the shifting cylinders of the conveyor.

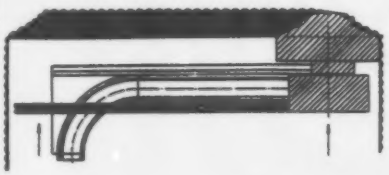


Fig. 2: The ESA-60-L starts to cut the coal over the full length of the working face.

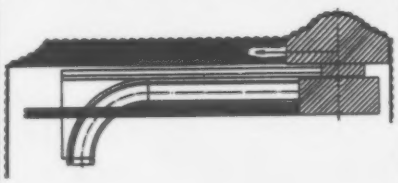
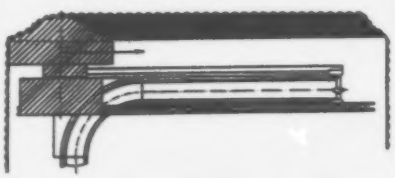


Fig. 3: If necessary, a cleaning up run is performed. The roof supports can then be advanced.



5.2 Interaction of Roof Supports and Conveyor in Face Openings and Roadway Headings

In general, only the advancing procedure of the shield support which is positioned at the EKF-0 tail end can be endangered. The danger in pulling the other two shields is less, because the conveyor is stabilized either by the two neighbouring shields or by the weight of the evacuating section.

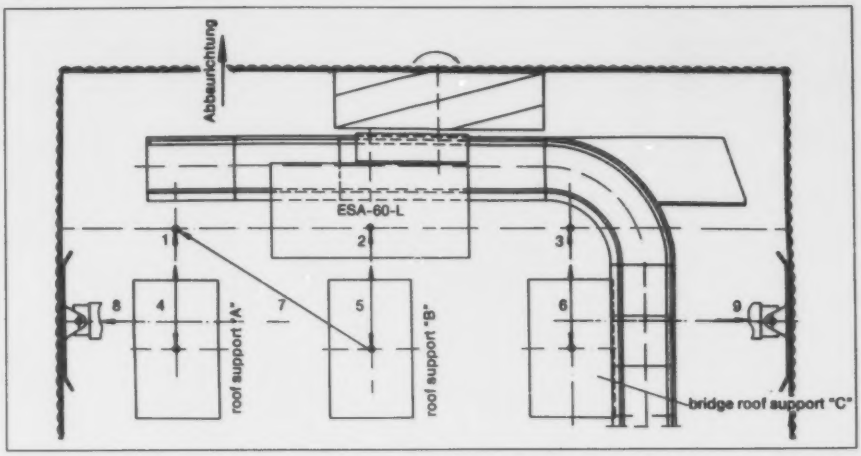
The transmission of high torques into the substructure of the conveyor and simultaneous stabilization can be effected by the horizontal arrangement of a diagonal cylinder from the centre shield to the conveyor substructure attachment point of the shifting cylinder of the shield to be pulled or by vertical anchoring of the conveyor on the face side by means of a perpendicular prop integrated in the substructure.

The operating cycle with diagonal anchoring can proceed as described by the following example: the schematically shown cylinders 1 – 3 serve for horizon control, cylinders 4 – 6 are the conveyor (roof support) shifting cylinders, cylinder 7 serves for diagonal anchoring, and cylinders 8 and 9 are provided for lateral anchoring.

Operating cycle:

Cutting

Cylinders 1 – 6 blocked
Cylinder 7 released
Cylinders 8 – 9 blocked



ESA-60-L operating cycle with diagonal anchoring

Advancing of the conveyor on the left

Cylinders 1–3	blocked
Cylinder 5	released
Cylinder 6	blocked
Cylinder 7	released
Cylinder 8	retracted
Cylinder 9	blocked
Cylinder 4	extended by shifting step

Advancing of the conveyor on the right

Cylinders 1–4	blocked
Cylinder 5	released
Cylinder 7	released
Cylinder 8	blocked
Cylinder 9	retracted
Cylinder 6	extended by shifting step

Pulling forward of shield 'A'

Cylinders 1–3	blocked
Cylinder 5	released
Cylinders 6–9	blocked
Cylinder 4	pulling shield 'A'

Pulling forward of shield 'B'

Cylinders 1–4	blocked
Cylinder 6	blocked
Cylinder 7	released
Cylinders 8+9	blocked
Cylinder 5	pulling shield 'B'

Shield 'C' accordingly

The ESA machines of the "L and K" series enable profiling of the roadway cross-section by simple means either manually or by an electronical control system.

5.3 Profile and Alignment Control

In roadway headings penetration into the face must naturally be effected in the coal or in the softest of minerals encountered, also making use of the cleavage and joint planes.

The roadway cross-section can be profiled by simple means either manually or by an electronical control system.

Manual Profile Control

A manual profile control system was first installed in a roadway heading at Zolder Colliery. It resulted from considerations of describing the different movements of the machine by a pattern of markings corresponding to the appropriate clearances. The drums of the 'L' series machines perform two different movements, i.e. the horizontal movement and the rotary movement, and the roadway profile is obtained by the interaction of these movements. Machines of the 'K' series are equipped with a further movement facility for lifting the machine body on one side. This increases the cutting range.

Manual Profile Control of the 'L' Series

The manual profile control system of the 'L' series will be shown and described by the example of a face heading using an ESA-60-L; ranging arm length 750 mm, drum diameter 1800 mm. The ranging arm is in the lower position and the machine is on the right-hand side. After shifting the conveyor on the left the ESA wedges itself into the face as usual whereafter the remainder of the conveyor is advanced.

The ESA now returns to the right-hand side according to a determined sequence of travelling movements and swivelling, clockwise movements of the ranging arm whereby the ranging arm performs a full revolution of 360°.

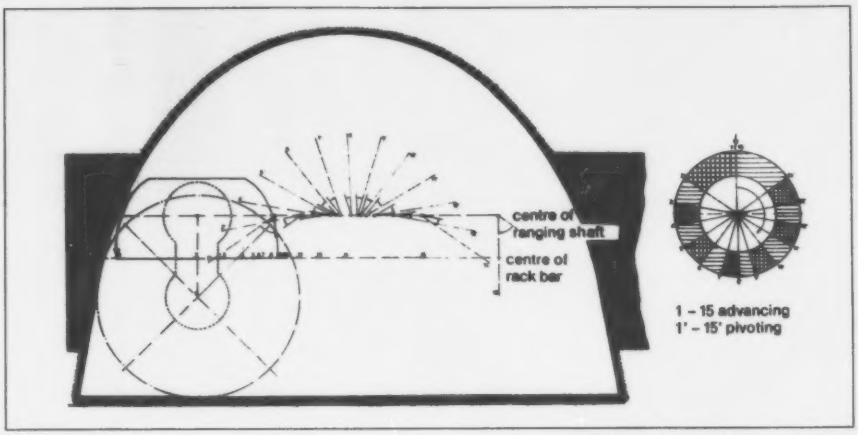
Cutting the profile with satisfactory precision is simplified by markings, in different colours, on both the rack bars and the ranging arm to which a disc is fitted for this purpose. The ESA travels over the full length of the white range (1 – 2). The ranging arm is then turned through a range which corresponds to the whole white marking on the disc (1' – 2'), and this procedure is successively repeated for each of the colours: travelling over the red range of the rack bar with subsequent turning of the ranging arm which corresponds to the red range etc. In this manner the profile is precisely cut on the condition that the conveyor was exactly horizontal and in the position of the roadway direction in the beginning. By cutting the whole profile in 24 stages a precision of 3 cm is reached.

As the contours of the roadway profile are defined as mathematical curves (the coordinates of the circle centres and of the radii are known according to DIN) the system is easy to apply.

Manual Profile Control of the 'K' Series

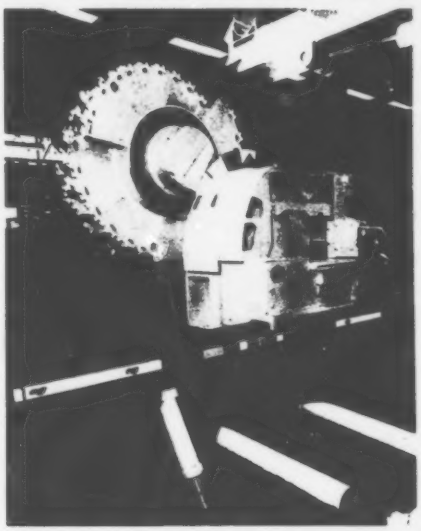
A manual profile control similar to the one for the 'L' series has been developed for the 'K' series.

The direction is controlled by a laser which is suspended in the roadway.

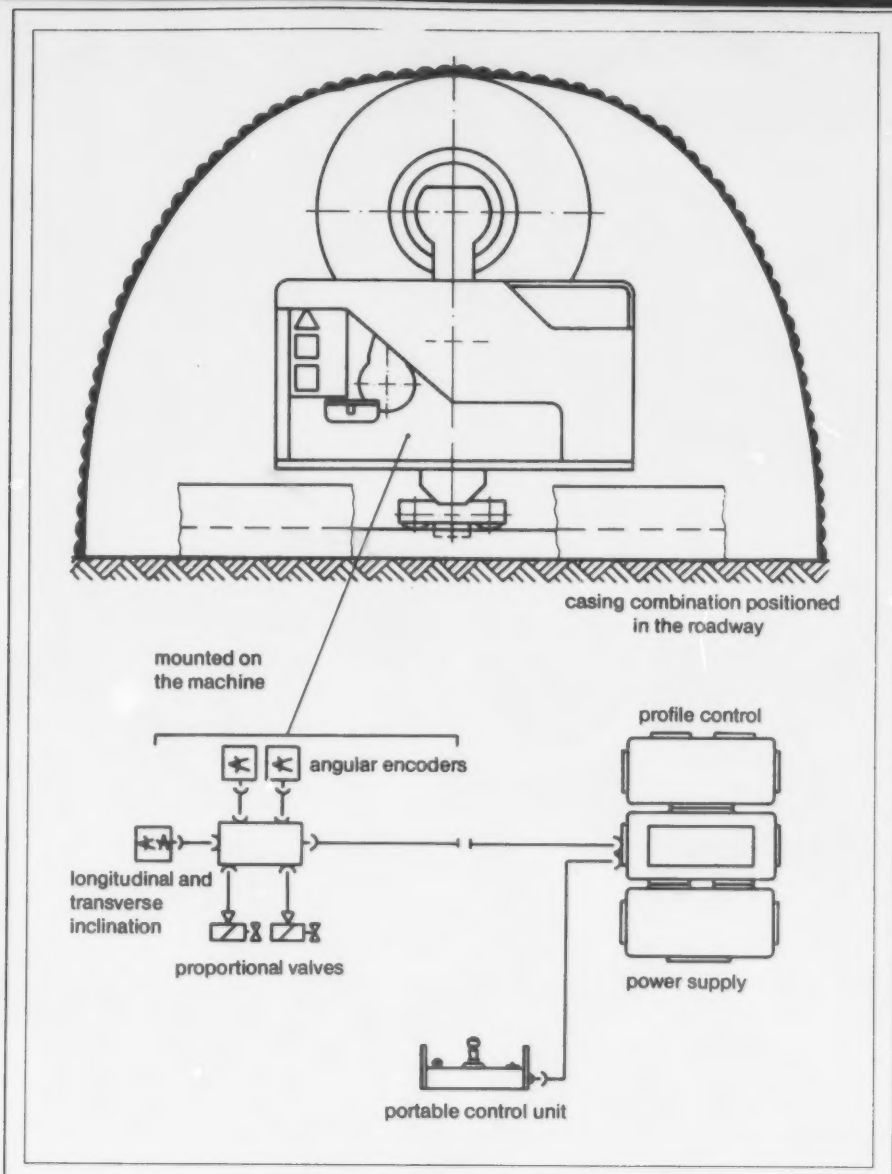


ESA-60-L Manual profile control

Profile and Alignment Control
The profile and alignment control allows for keeping the heading equipment properly aligned and for automatically guiding the cutting drum along the profile contour.



ESA-60-L with coloured marking pattern for profile control



ESA-60-L Profile control

Profile Control

Machines of the ESA-60-/150-L series can perform two movements, i.e. travelling and swivelling. Both positions are detected by angular encoders and together with the values of the longitudinal and transverse inclination of the conveyor they are fed into a computer where they are compared in a ROM memory with the given nominal values for the profile contours. The computer

Profile control parameters

output controls proportional valves for the drum height and the haulage speed.

The following modes of operation are possible:

1. Manual control: the computer does not interfere; the drum can cut beyond the profile contours.
2. Semi-automatic operation: the computer prevents cutting beyond the profile contours.
3. Automatic operation: the computer automatically controls the re-profiling cut.
4. Program: the cutting drum operates according to a program.

Alignment Control

Alignment control is ensured by a laser which is suspended in the roadway. Angular and parallel deviations are displayed on the heading equipment and the necessary corrections are manually fed in, in the decimal system, by means of coding switches.

Computer control is ensured by a microprocessor of the CMOS technique with the corresponding input and output values and memories. The system complies with the protection class for intrinsical safety.

Wide face openings require a starting section of which only 7 m must be manually driven.

6. Description of Some Installations

6.1 ESA in Face Openings

6.1.1 Face Opening over the Full Width

6.1.1.1 Start of a Face Opening

Wide face openings to be driven by the ESA require a starting section which is produced in four phases before the regular drivage can start.

After the installation of the roadway conveying means which generally consist of a stage loader and a belt conveyor, about 2 m must be manually driven in the conventional manner, so that the EKF-0 curved conveyor can be installed.

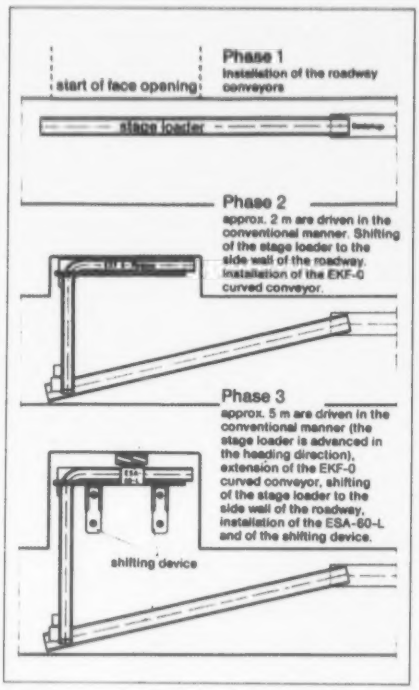
After another 5 m of conventional drivage the EKF-0 conveyor has reached its final length. At the same time the ESA is mounted on to the accessories. During this first phase the roadway stage loader moves from the wall opposite to the face opening in the heading direction, thus following the drive head of the EKF-0 conveyor. This offers the advantage that the face opening can be advanced by the width of the roadway without an extension of the conveyor.

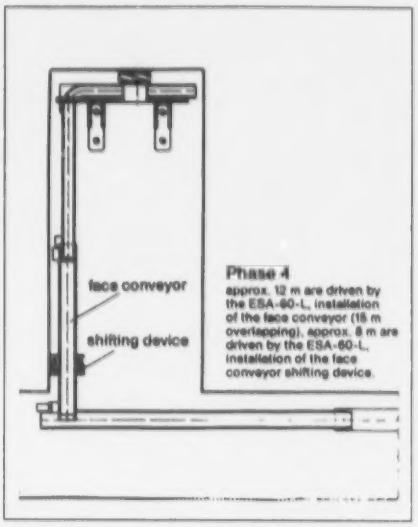
When the face opening has been driven manually over a length of only 7 m the ESA takes over and drives initially 12 m and then another 8 m, so that the face conveyor with an overlapping length of 15 m as well as the face conveyor shifting device can be installed in the meantime.

The regular heading work now commences in the face opening up to the intersection.

Only 7 m of the total length of approx. 27 m of the starting section must be driven manually.

The following illustration shows the various phases of the starting situation of a face opening with the ESA-60-L .


start of face opening
Phase 1
Installation of the roadway conveyors
stage loader
Phase 2
approx. 2 m are driven in the conventional manner. Shifting of the stage loader to the side wall of the roadway. Installation of the EKF-0 curved conveyor.
Phase 3
approx. 5 m are driven in the conventional manner (the stage loader is advanced in the heading direction), extension of the EKF-0 curved conveyor, shifting of the stage loader to the side wall of the roadway, installation of the ESA-60-L and of the shifting device.
shifting device


face conveyor
shifting device
Phase 4
approx. 12 m are driven by the ESA-60-L, installation of the face conveyor (18 m overlapping), approx. 8 m are driven by the ESA-60-L, installation of the face conveyor shifting device.

Start of face opening with the ESA-60-L

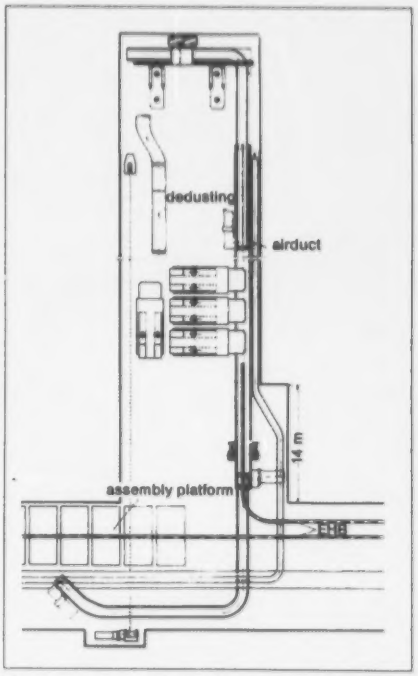
6.1.1.2 Face Opening and Simultaneous Installation of Shield Supports at Osterfeld Colliery

In March/April 1985 a face opening was driven in the Zollverein 1 seam at Osterfeld Colliery. The face opening had a width of 7,70 m and a height of 1,90 m. When a face opening length of 71 m was reached the shield supports were installed without the heading work being interrupted.

According to the organization program the first 15,50 m were worked by hand and the first components of the equipment were then installed within 5 days.

After the installation of the ESA accessories, the ESA itself, the forepoling units, the shifting device, and part of the final face conveyor the ESA started with the heading operation up to 28,5 m. The conveyor pushing device was then installed and the drivage of the 300 m long face opening continued.

The shield supports were pre-



Face opening and simultaneous installation of shield supports

The face opening from the base road at Osterfeld with the ESA-60-L increased the per-

assembled in the roadway and a plough drive pulled them into the face opening.

Within 24 days 284,5 m were driven with the ESA-60-L up to the intersection in 4/3 shifts a day. The average advancing speed amounted to 11,85 m/d, and a maximum daily advance of 17,25 m was reached.

6.1.2 Face Opening from a Base Roadway at Osterfeld Colliery

Face openings starting from a base roadway offer the following advantages:

- Due to the extraction of the coal up to the roadway end there are no pillars left which would have a negative effect on other workings later on.
- If the roadway is to be maintained a dam can be built immediately behind the heading equipment in the face opening.
- The shield supports can be installed more easily than in a wide face opening while drivage continues.
- The evacuation of the material cut is independent of the installation of the shield supports.
- Access into the face opening from the side of the base roadway is easy.

Such a face opening with the ESA-60-L is shown by the example of the Zollverein 1 seam with a seam thickness of 1,2 m and a cutting range of 1,6 m at Osterfeld Colliery.

In view of the future mining operation it was necessary to keep the base roadway in as good a condition as possible over the full running time of the panel. The stability of the roadway was ensured by a dam of 1,5 m in width built approx. 6 m behind the ESA-60-L.

The ESA-60-L was mounted on an EKF-0 conveyor positioned at right angles to the direction of advance and the material cut was discharged on to an EKF-III conveyor in the roadway.

The wedge cutting method usually applied in face openings with the ESA-60-L was avoided, because the machine could travel into the base roadway, so that the drum could attack the face with its full depth.

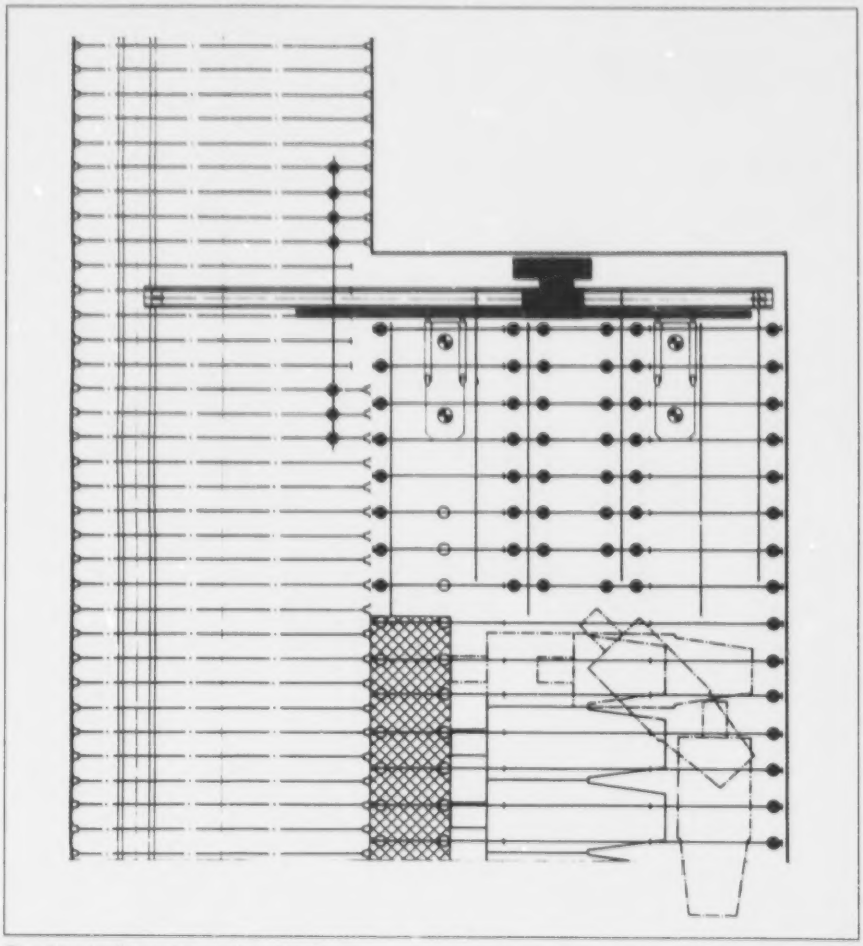
The shield supports were transported through the future haulage roadway into the face opening and were installed on the same level as the end of the dam.

This application of the ESA-60-L offers further advantages:

- No starting section is required for the assembly of the ESA.
- Easy maintenance of the ESA, because it can travel into the base roadway.
- The increase in performance amounted to more than 30% as compared with conventional face openings.

The advancing rate amounted to approx. 60 cm/MS as compared to 25 cm/MS in conventional face openings.

formance by more than 30 % as compared with conventional face openings.



The illustration shows the situation of the face opening

Face opening from a base roadway

The ESA system has been successfully used in the face end area at the collieries Fürst Leopold, Nordstern and

6.2 ESA in Stableholes and Gate Roads

Possibilities of Application

The tasks of a mining machine at the face ends can be defined as follows:

– Tasks of the equipment

(i.e. the drivage of those workings which are required for exposing the face for the scheduled mining operation)

These include:

- Roadways driven in line with the face
- Roadways lagging behind the face
- Advanced roadway headings.

– Tasks of the Mining Organisation

These include:

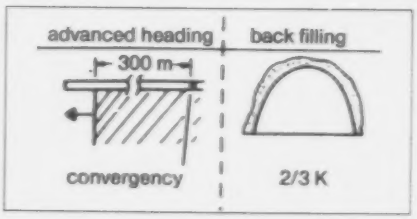
- Provision of a new starting point for the shearer loader on the face in order to avoid shunting or the semi-face method which are time consuming.
- Provision of a new starting point for the plough on the face (cutting of the plough stablehole).

This results in a number of possible combinations at the tail gate and at the main gate which can be solved by mounting the machine on separate or on the face conveyor accessories.

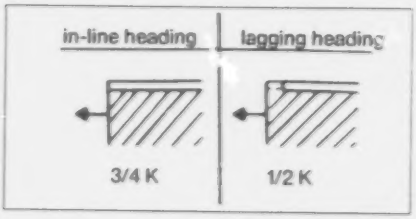
The drivage of the roadway in line with the face requires special coordination of the equipment installed and of the operating cycle. The ESA must adapt itself to the working cycle of the production machine in order to avoid unnecessary idle times; at the same time the cross-section of the roadway must be cut as quickly and as precisely as possible in order to gain time for setting the roof supports.

The convergency behaviour is different in advanced, in-line or lagging roadway headings; it is schematically shown in the following illustration.

In advanced roadway headings the whole of the subsidence pressure ahead of the face has a disadvantageous effect on the convergency behaviour. Roadways driven in-line with the face are more favourable with regard to rock mechanics, as the leading pressure has no effect. Lagging roadway headings have a greater stability as even



Influence of roadway headings on the convergency



Westerholt as well as in Belgium at Zolder, Waterschei and Winterslag.

part of the trailing pressure does not affect them. The advantages of less convergency allow for the choice of smaller cross-sections which has a positive effect on the roadway heading costs.

The ESA system has been successfully used in the face end area at the collieries Fürst Leopold, Nordstern, and Westerholt as well as in Belgium at Zolder, Waterschei, and Winterslag.

In the Belgian mines the ESA is used as a face end system on shearer loader or plough faces as well as in advanced roadway headings.

The tail gate roads of two plough faces in level and inclined seams were driven at the collieries Nordstern and Westerholt.

At Zolder Colliery the ESA-60-L and ESA-60-K were installed in advanced roadway headings and haulage roadways.

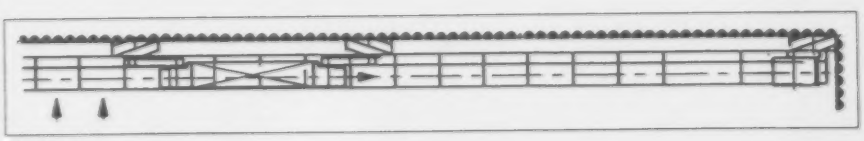
6.2.1 Roadways Driven in Line with and Lagging Behind the Face

6.2.1.1 Double-ended Ranging Drum Shearer Loader and ESA at the Face End at Zolder Colliery

At Zolder Colliery the tail gate roads of several shearer loader faces in moderately inclined seams are driven in line with the face by means of the ESA-60-L. The face advance is thereby increased, because the double-ended ranging drum shearer installed starts on the downward traverse immediately after the completion of the upward traverse, so that the time required for an upward traverse and a downward traverse is reduced. The time-consuming shunting at the tail gate is avoided.

The interaction between the double-ended ranging drum shearer and the ESA-60-L is represented by means of the CAD process sequence program. In such a configuration the cutting drum diameter of the ESA-60-L should correspond to the seam thickness.

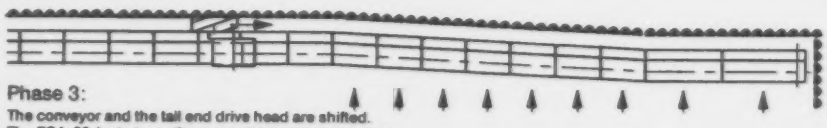
The shield supports are naturally operated according to the one web back system which means that immediately after the passage of the machine the roof supports are advanced first and then the conveyor is shifted.



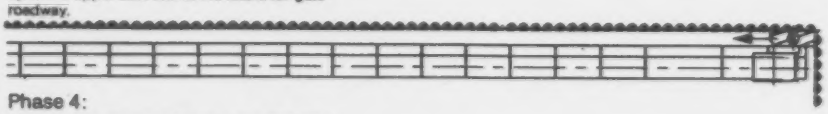
Phase 1:
The double-ended ranging drum shearer is on the upward traverse and travels into the section cut by the ESA-60-L. The conveyor is shifted immediately behind the double-ended ranging drum shearer

Passing over the tail end drive head:
Power loaders
- double-ended ranging drum shearer
- ESA-60-L (seam thickness = drum diameter)

Phase 2:
Both the double-ended ranging drum shearer and the ESA-60-L start on the downward traverse.



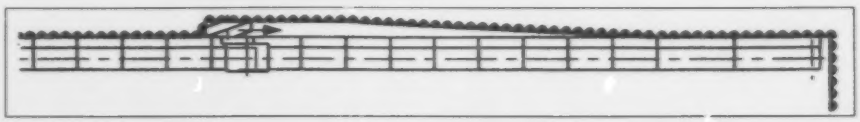
Phase 3:
The conveyor and the tail end drive head are shifted.
The ESA-60-L starts on the upward traverse and travels up to the upper side wall of the future tail gate roadway.



Phase 4:
After pivoting of the cutting drum the ESA-60-L starts cutting the step left on the floor.

Phase 5:
The conveyor is shifted.
The ESA-60-L wedges itself into the face.

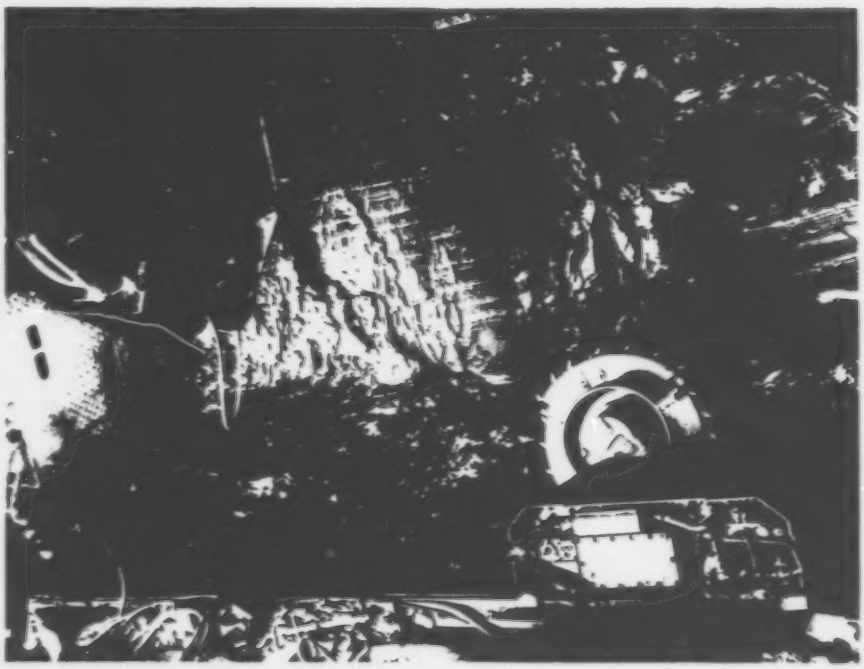
Phase 6:
The conveyor and tail end drive head are shifted.



Phase 7:
The ESA-60-L starts cutting the remaining wedge. (The operational sequence in the future roadway area corresponds to that of phase 4).

6.2.1.2 Tail Gate Road Driven in Line with a Plough Face at Nordstern Colliery

An ESA-60-L with a 750 mm long ranging arm and a 1500 mm dia. cutting drum is installed at the colliery Nordstern/ Zollverein as a face end machine in the tail gate of a face in the Gustav seam.



Situation in the tail gate

The production machine on the face is a plough.

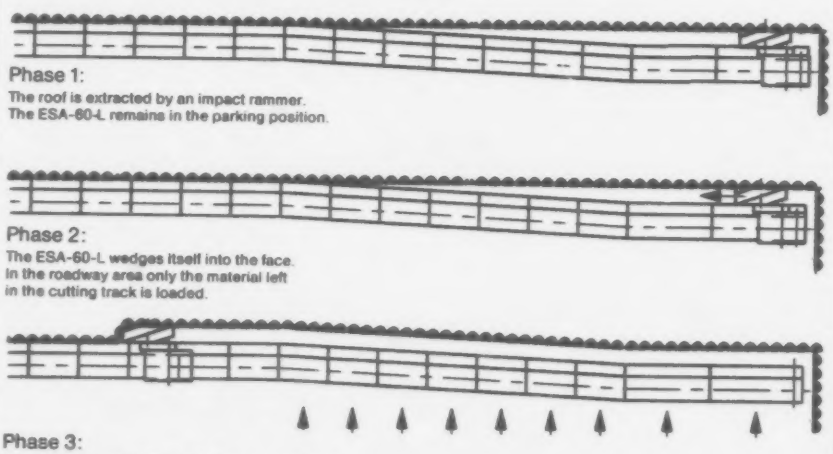
The ESA-60-L serves for producing a new point of attack for the sliding plough approx. 25 m below the tail gate and to cut part of the 24 m^2 roadway profile.

Further machines installed in the tail gate road include an impact rammer and a dinting loader. Moreover, a pneumatic stowing pipe is installed at the lower side wall.

Due to the concentration of equipment in this area the initial cross-section of 19 m^2 had to be increased to 24 m^2.

The seven phases of the interaction between the ESA-60-L and the sliding plough are shown in the following illustration.

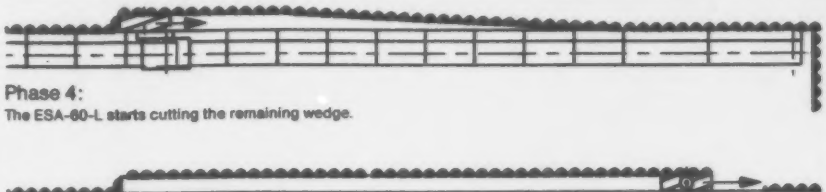
Passing over the tail end drive head:
winning machines
– plough
– ESA-60-L (seam thickness = drum diameter)



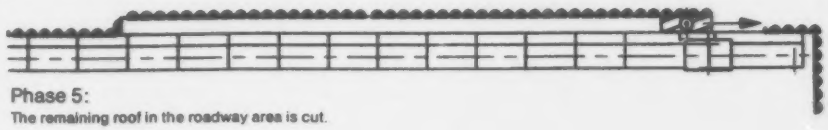
Phase 1:
The roof is extracted by an impact rammer.
The ESA-60-L remains in the parking position.

Phase 2:
The ESA-60-L wedges itself into the face.
In the roadway area only the material left in the cutting track is loaded.

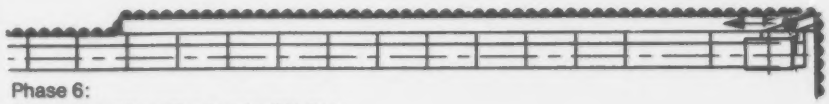
Phase 3:
The conveyor and tail end drive head are shifted.



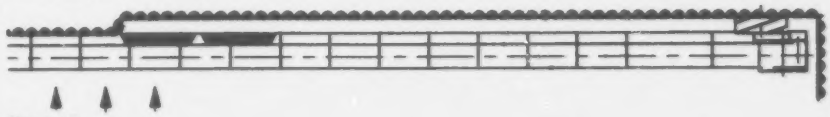
Phase 4:
The ESA-60-L starts cutting the remaining wedge.



Phase 5:
The remaining roof in the roadway area is cut.



Phase 6:
After pivoting of the cutting drum the ESA-60-L starts cutting at the floor in the roadway and returns immediately into the parking position.



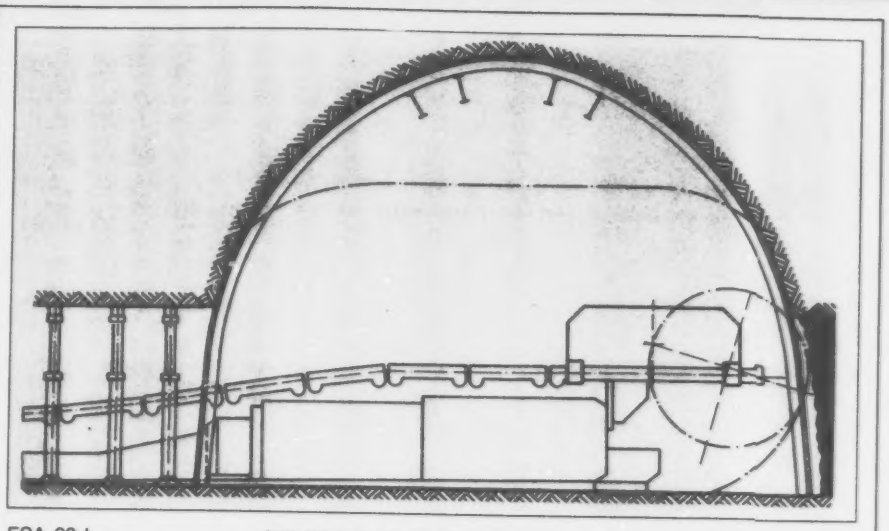
Phase 7:
The plough now has a new point of attack. Immediately after the passage of the plough the conveyor is shifted.

The working sequence in the tail gate starts with the extraction of the rock at the roof by means of the impact rammer which offers the advantage that a slope is obtained in the cutting web, so that the remaining material extracted is automatically loaded on to the conveyor. During this time the ESA-60-L is positioned at the upper side wall of the tail gate. The cantilever rails are then pushed forward and the pre-assembled arches are set by means of a roof support setting aid which is bolted to the pick of the impact rammer (arch spacing 0,80 m). The impact head is provided with a supporting platform.

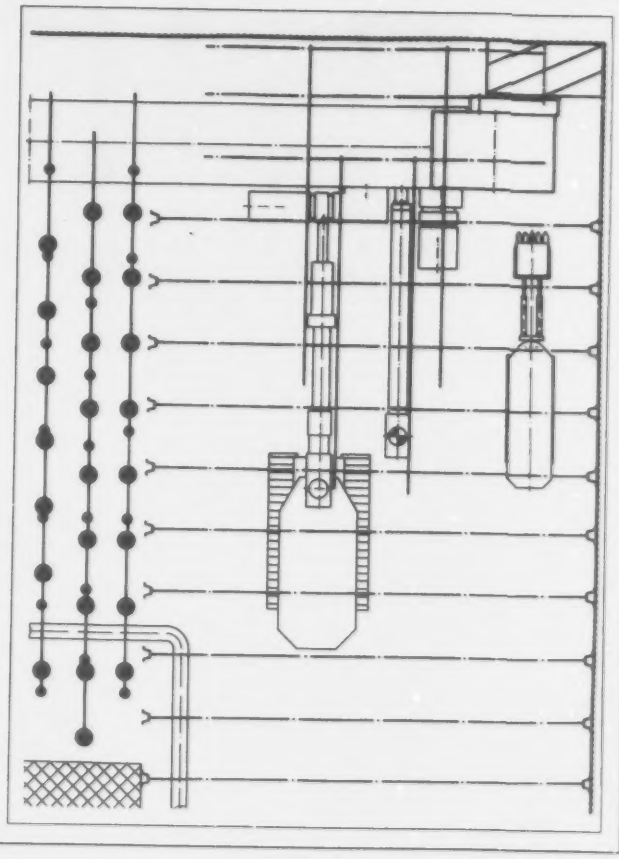
The ESA-60-L now wedges itself into the face, but first it must load the material from the slope. The dinting loader serves for loading the remaining material from the upper side wall on to the conveyor.

The roof in the cutting range of the ESA-60-L is extracted after the completion of the upward traverse. The following illustrations show the situation in the tail gate road.

The installation proved that the ESA-60-L achieves the tasks at the face end. However, it has also shown that only one machine should be used for cutting the full roadway cross-section in order to avoid a concentration of equipment at the face end. As a consequence an ESA-60-K or an ESA-150-K could be installed by means of which the roof can be exposed up to a height of about 5 m.



ESA-60-L –
tail gate roadway
driven in line with
the face



ESA-60-L – tail gate
roadway driven in line
with the face. Situation
at the working face

Situation in the tail gate roadway

6.2.1.3 Combined In-line and Lagging Drivage of the Tail Gate Road of a Steeply Inclined Plough Face at Westerholt Colliery

The following example describes the installation of an ESA-60-L in the face end area and in the tail gate road of a plough face in an inclined seam at Westerholt Colliery. Seam Q is inclined from 37 to 40 gon. The ESA is retained on the inclination by means of independent braking systems.

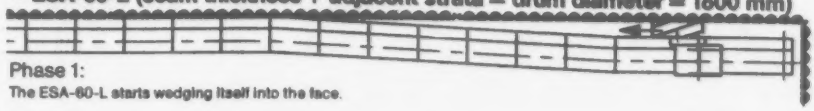
The ESA-60-L is used for cutting the roof in the tail gate true to profile. The remaining step in the floor is conventionally removed about 20 m behind the face.

The interaction between the ESA-60-L and the sliding plough as well as the operating sequence of the ESA can be represented in seven phases.

This installation has proved the operational safety of the ESA in steeply inclined seams. However, due to the difficult roof condition at the face end the total cross-section should be cut in future.

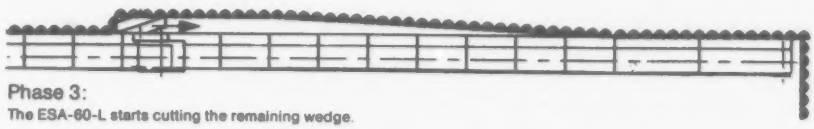
Moreover, cross-sections illustrate the situation in the roadway heading, at the face end and approx. 20 m behind the face. Roof support setting is also shown.

Passing over the tail end drive head
– plough
– ESA-60-L (seam thickness + adjacent strata = drum diameter = 1800 mm)



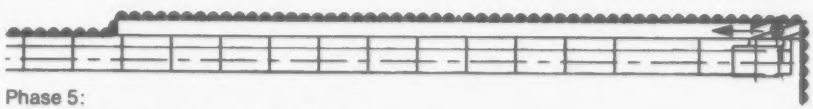
Phase 1:
The ESA-60-L starts wedging itself into the face.

Phase 2:
The conveyor and tail end drive head are shifted.

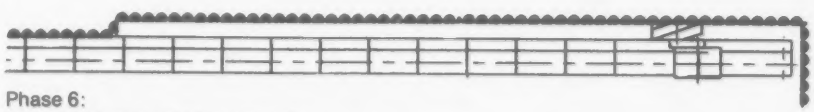


Phase 3:
The ESA-60-L starts cutting the remaining wedge.

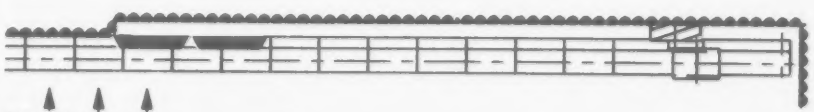
Phase 4:
In the roadway area the roof is cut true to profile.



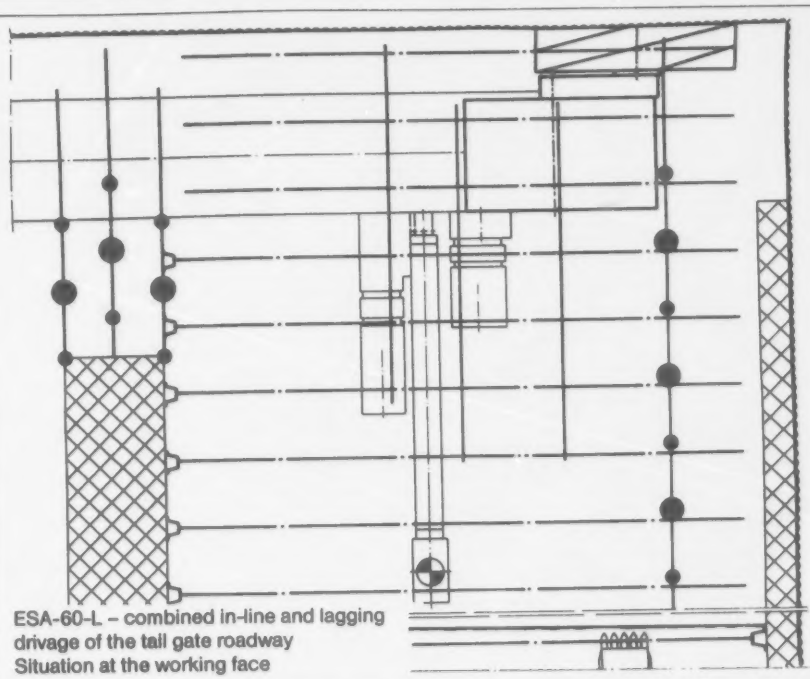
Phase 5:
After pivoting of the cutting drum the ESA-60-L starts cutting at the floor.



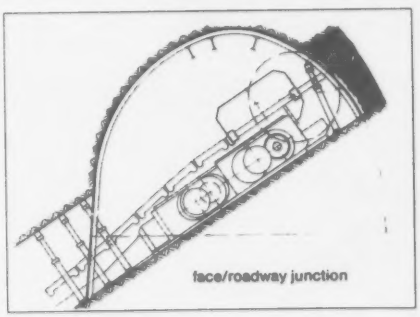
Phase 6:
The ESA-60-L stops at the face entry and remains in the parking position.

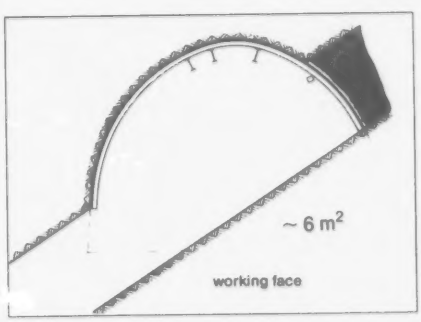


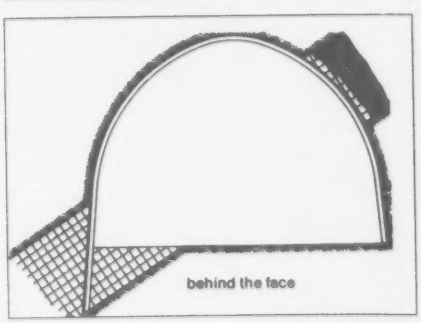
Phase 7:
The plough now has a new point of attack. Immediately after the passage of the plough the conveyor is shifted.



ESA-60-L – combined in-line and lagging drivage of the tail gate roadway
Situation at the working face







6.2.2 Advanced Tail Gate Roadway Heading at Zolder Colliery

The advantages of advanced roadway headings include the decongestion and hence the relative independence of the roadway heading from the face operations. A major disadvantage could occur due to a more important convergency in advanced roadway headings as compared with in-line or lagging roadway headings. Comprehensive dinting might be required close behind the roadway heading.

With regard to the choice of distance to the face a minimum of 15 to 20 m has proved to be favourable on account of the subsidence pressure ahead of the face, but in connection with this the local conditions must be particularly taken into account.

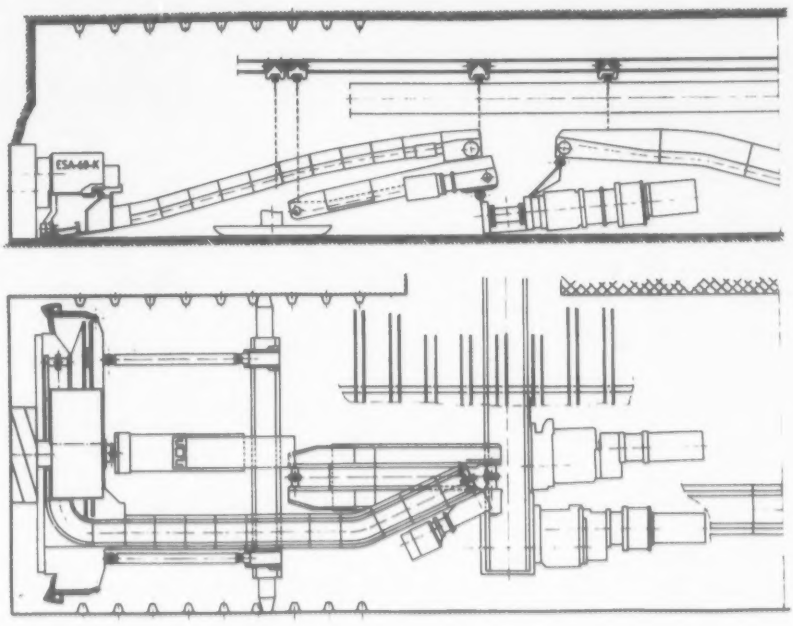
As an example for an advanced tail gate roadway heading the installation of an ESA-60-K at Zolder Colliery will be described in the following. Initially the ESA-60-K preceded the plough face by 9 m and later on, when a better convergency control was achieved, by 15 m. In the first panel it was only necessary to cut the roadway cross-section of 14 m^2 true to profile with an advance of 3 m/d corresponding to the face advance. The plough stable was manually excavated.

The mechanical equipment in the roadway area is shown on the following illustrations:

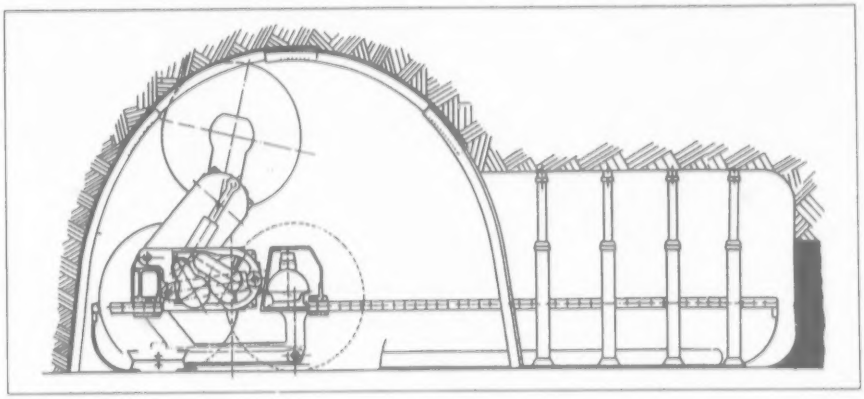
- heading conveyor with additional 45° curve;
- intermediate conveyor the length of which corresponds to the daily heading advance (and the movability of the ESA) (approx. 5 to 6 m);
- stage loader of a following dinting area;
- face conveyor with tail end drive head;
- roof support setting aid with lateral bracing.

In the heading two shifts are worked. The cross-section is cut with the aid of a manually controlled coloured profile pattern.

In the second panel the ESA-60-K had the additional task of cutting the plough stablehole with a width of 2,5 m and a height of 1,8 m at the same time as driving the roadway heading. The stablehole area is supported by single props with Van Wersch roof bars.



ESA-60-K in an advanced roadway heading



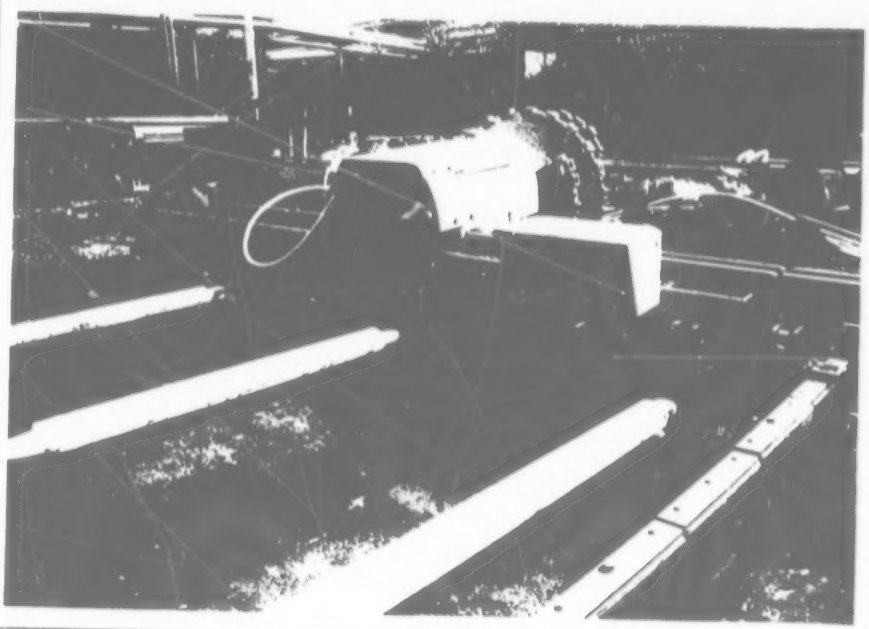
Lateral bracing is ensured by the extension of the corresponding cylinder by 2,5 m in a double telescopic manner. Before the heading equipment is advanced the cylinder is retracted into the roadway to prevent its collision with the roadway roof supports already set.

The ESA-60 used as a stable-hole machine increases the face advance and facilitates the strenuous physical work in the stablehole area.

6.2.3 Cutting of the Stablehole at Fürst Leopold Colliery in Wulfen

At Fürst Leopold Colliery in Wulfen the stablehole at the tail end drive head of a plough face in the Hagen seam was cut by means of an ESA-60. The bad condition of the roof support face segments in the tail gate necessitated the positioning of the tail end drive head of the face conveyor within the 1,30 m thick face. The conventional preparation of the stablehole only enabled a maximum face advance of 6 m/d with a crew of 25 men. After the installation of the ESA-60 as a stablehole machine a face advance of approx. 8 m/d and daily outputs of 3000 t of coal (maximum 3500 t of coal) were reached during three-shift operation.

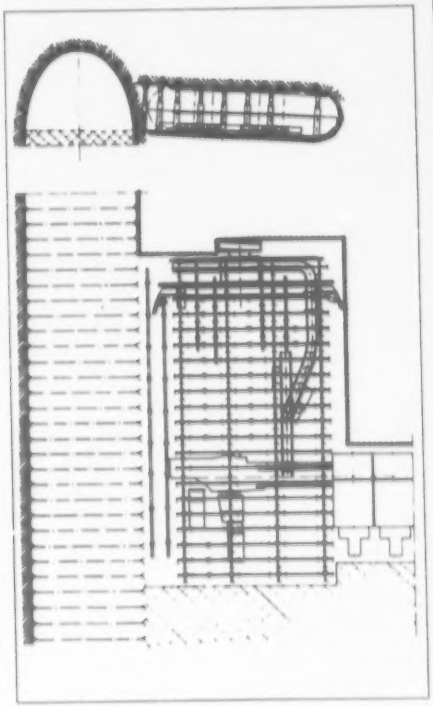
The stablehole has a width of 6,50 m and a height of 1,60 m. In order to facilitate penetration into the face the ESA is equipped with a sumping-in drum. As usual the conveyor installed with the



ESA machine has a 90° curve and appropriately overlaps an intermediate conveyor, so that the ESA is independent of the face operations for a short period of time.

The larger space requirement in comparison to conventional stableholes and consequently the greater roof surface to be supported (ESA: 110 m^2/conventional: 60 m^2) was met by setting cross-type and Van Wersch roof bars.

This system has contributed to a further mechanization of the face end area. It not only enabled an increased face advance but also rendered the strenuous work in the stablehole area more human.



ESA-60-L/150-L shearers are economic on shortwall faces up to a length of 50 m due to their minimal space requirement, compact design, and easy assembly.

6.3 Mining Method

6.3.1 Shortwall Faces using Stolzenbach Colliery as an Example

The ESA-60-L or ESA-150-L allow for mining shortwall faces up to a length of approx. 50 m economically. The limit is set by the smaller area exposed due to the slower travelling speed and the narrower web as compared with double-ended ranging drum shearer loaders.

Economical operation of the ESA on shortwall faces is ensured by the minimal space requirement as well as the quick and easy assembly and transportation in the case of frequent face moves.

Shortwall face mining is widely applied in Eastern European countries.

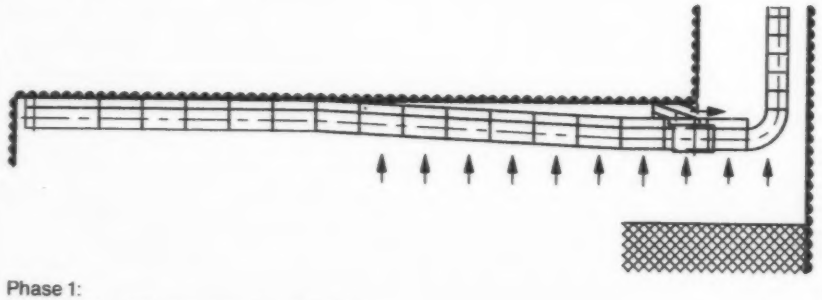
The sub-level method is frequently applied for mining thick seams, but shortwall faces on which the roof coal is drawn off are also widely worked.

Moreover, the shortwall method is applied for the extraction of residual pillars or as an alternative for board-and-stall workings. The working cycles for advancing or retreating shortwall faces are shown on the following CAD illustrations.

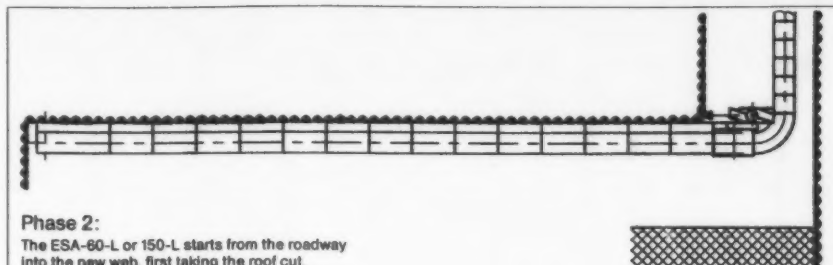
Since 1982 the ESA-60-L is successfully installed in the soft lignite underground

Shortwall face
(single entry retreating face)
Power loader:
ESA-60-L or 150-L
(seam thickness greater than drum diameter)

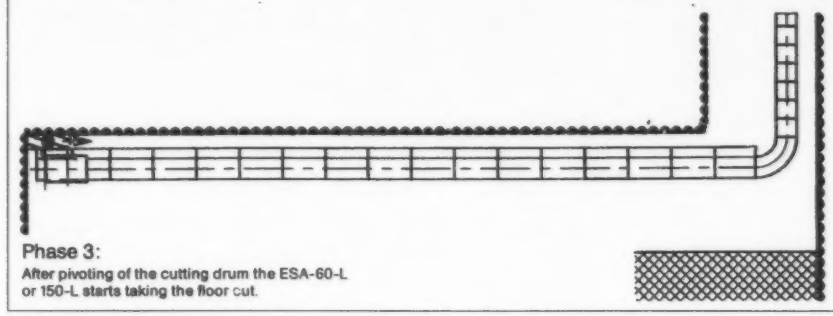
Conveyor:
Discharge end or curve



Phase 1:
The ESA-60-L or 150-L passes over the drive head after having taken the floor cut. The conveyor is then shifted.



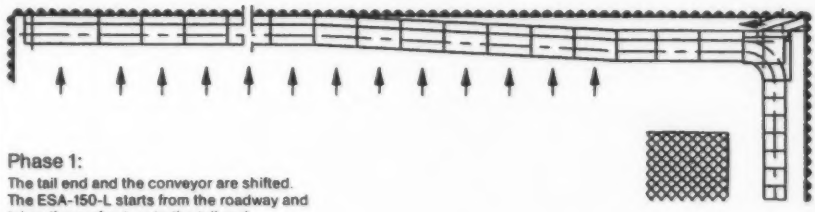
Phase 2:
The ESA-60-L or 150-L starts from the roadway into the new web, first taking the roof cut.



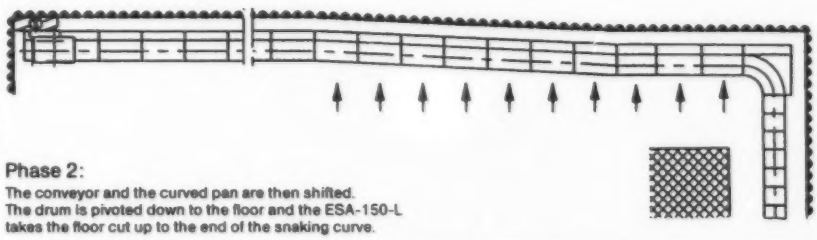
Phase 3:
After pivoting of the cutting drum the ESA-60-L or 150-L starts taking the floor cut.

Shortwall face
(single entry advancing face)
Power loader: ESA-150-L
(seam thickness greater than drum diameter)

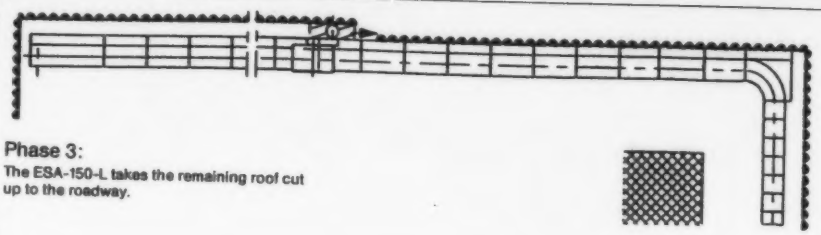
Conveyor: width approx. 632 mm
Curved pan or roller curve



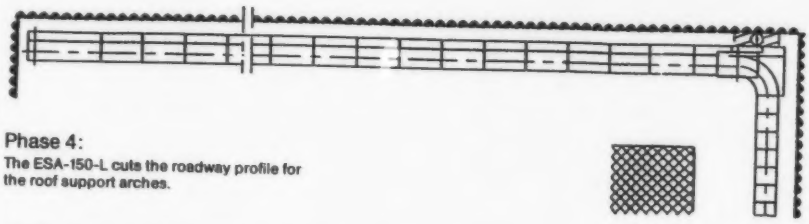
Phase 1:
The tail end and the conveyor are shifted.
The ESA-150-L starts from the roadway and takes the roof cut up to the tail end.



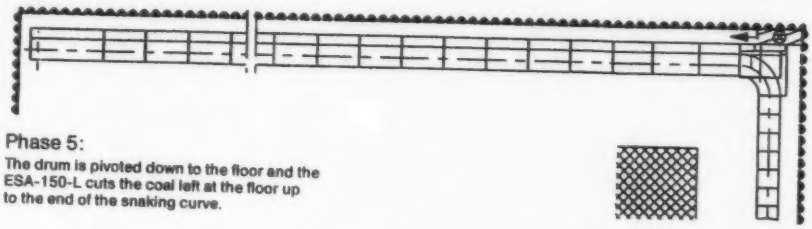
Phase 2:
The conveyor and the curved pan are then shifted.
The drum is pivoted down to the floor and the ESA-150-L takes the floor cut up to the end of the snaking curve.



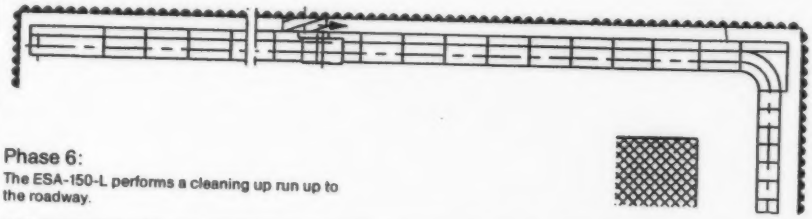
Phase 3:
The ESA-150-L takes the remaining roof cut up to the roadway.



Phase 4:
The ESA-150-L cuts the roadway profile for the roof support arches.



Phase 5:
The drum is pivoted down to the floor and the ESA-150-L cuts the coal left at the floor up to the end of the snaking curve.



Phase 6:
The ESA-150-L performs a cleaning up run up to the roadway.

mine Stolzenbach. In 14 panels with face lengths from 30 to 50 m average advancing speeds of up to 7 m/shift and outputs up to 1400 t/shift have been reached up to now. A modified working method of the winning machine is applied due to the poor roof conditions and the friable coal. From the roadway the ESA only cuts along the floor and then performs the loading traverse. The shield supports are advanced one step forward while being in contact with the roof, and the roof coal is extracted, if required, by the extension of the sharpened sliding roof bar.

In 14 panels with 30 to 50 m long faces average advancing speeds of up to 7 m and outputs up to 1400 t per shift have been reached.

6.3.2 Alternative Mining Methods

Worldwide research is being carried out in view of the economical extraction of coal from areas which normally do not lend themselves to longwall mining. The reasons for these efforts are, for instance, the extraction of residual pillars necessary due to rock mechanics, mining damage considerations, or the fact that geological disturbances would entail layout problems in the case of longwall mining.

A proposal of the Mining Research and Development Establishment of the NCB to render the performance of a less productive longwall system competitive and to improve it in some cases involves the ESA single entry mining system.

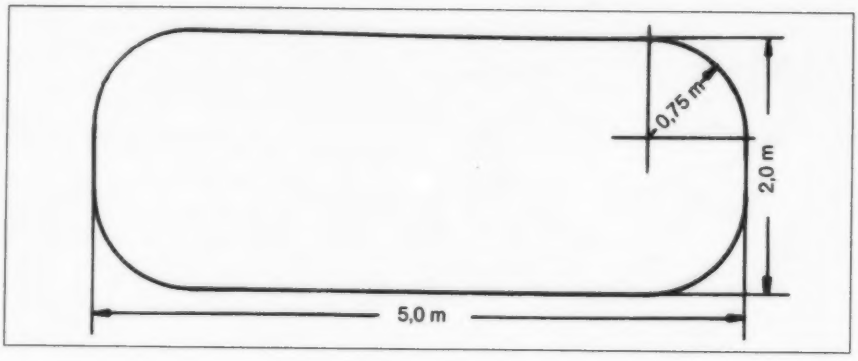
The proposal is aimed at reducing the expense of material and personnel whilst simultaneously ensuring outputs of 250 – 300 t/shift.

The system comprises four phases:

Phase 1:

- Drivage of a 500 – 600 m long in-seam roadway with a minimum cross-section (e.g. 5 m x 2 m) and line supports
- Installation of a ventilation system for the roadway heading, the face opening and the retreating face
- Installation of the manriding and material handling equipment.

Calculation of performance:



Roadway cross-section

$$A = 5\,m \cdot (2\,m - 1{,}5\,m) + 2 \cdot 0{,}75\,m\,(5\,m - 1{,}5\,m) + 0{,}75^2\,m^2 \cdot \pi$$

$$= \underline{\underline{\mathbf{9{,}52\ m^2}}}$$

Specific output

$$m = A_{Gew} \cdot s_{FL}$$

$$= 9{,}52\,m^2 \cdot 1{,}3\,\frac{t}{m^3}$$

$$= \underline{\underline{\mathbf{12{,}37\ \frac{t}{m}}}}$$

Planned output:

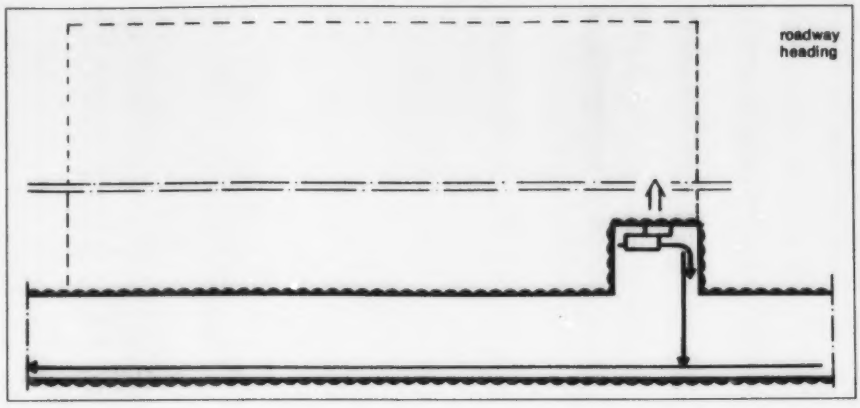
$m = 250 \ldots 300$ t/shift

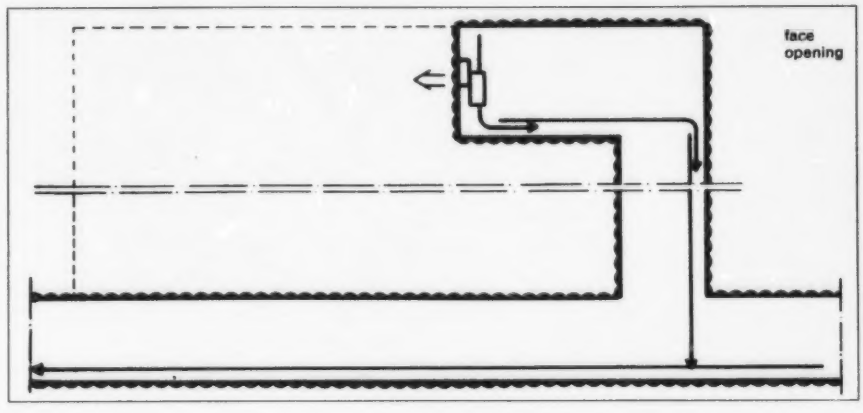
Heading advance

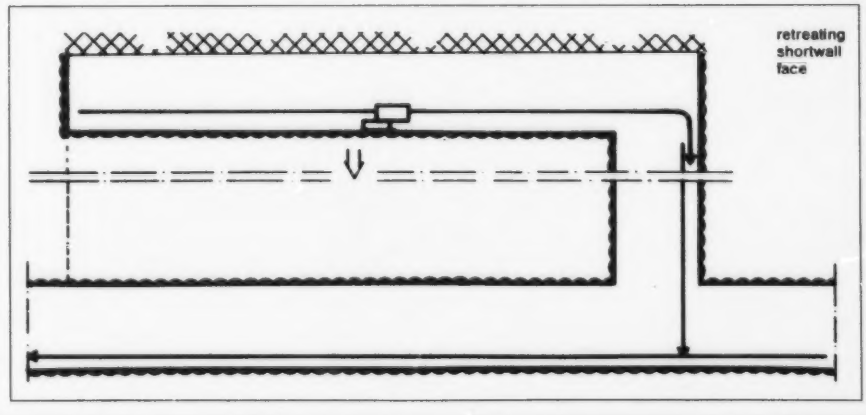
$$V = \frac{m}{m} = \frac{250 \ldots 300\ t/shift}{12{,}37\ t/m}$$

$$= \underline{\underline{\mathbf{20{,}21 \ldots 24{,}25\ m/shift}}}$$

Single Entry Mining System



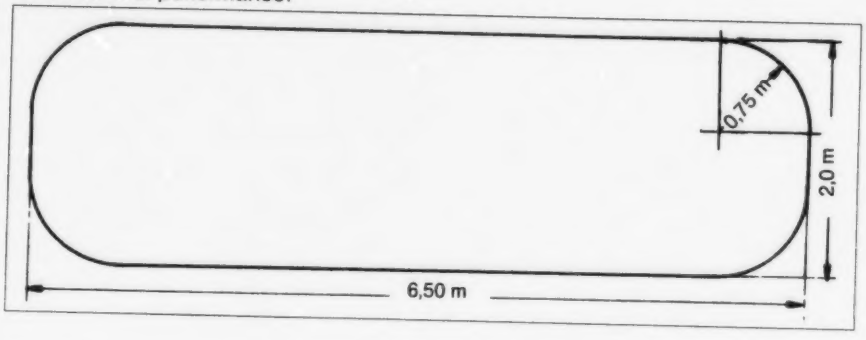




Phase 2:

- Preparation of the approx. 5 m long starting area for the face opening some meters before the end of the roadway
- Turning of the ESA equipment and installation of a further line pan into the EKF-0 heading face conveyor
- face opening over a length of 40 m with a cross-section of 6,50 m x 2 m and line supports (for detailed sequence: see item 6.1 – ESA-60-L in face openings)
- Extension of the airduct
- Installation of a belt conveyor in the roadway

Calculation of performance:



Cross-section of face opening

$$A = 6{,}5\ m\ (2\ m - 1{,}5\ m) + 2 \cdot 0{,}75\ m\ (6{,}5\ m - 1{,}5\ m) + 0{,}75^2\ m^2 \cdot \pi$$

$$= \mathbf{12{,}52\ m^2}$$

Specific output

$$m = A \cdot s$$

$$= 12{,}52\ m^2 \cdot 1{,}3\ t/m^3$$

$$= \mathbf{16{,}27\ t/m}$$

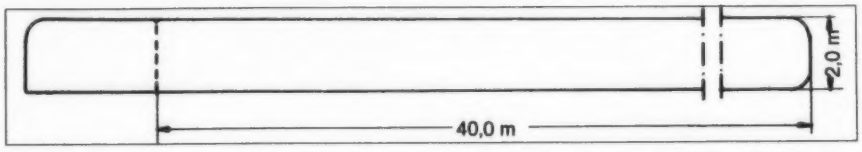
Heading advance:

$$v = \frac{m}{m} = \frac{250 \ldots 300\ t/shift}{16{,}27\ t/m}$$

$$= \mathbf{15{,}36 \ldots 18{,}44\ m/shift}$$

Phase 3:

- Mounting of the ESA-60-L on to the face accessories
- Installation of the shield supports
- Removal of the face opening support
- Displacement of the airduct to the goaf side of the conveyor



Phase 4:

- Retreat mining of the shortwall face (for detailed sequence: see item 6.3 – Mining Method)
- Removal of the roadway supports

Calculation of performance:

2,0 m 40,0 m

$A = 80\ m^2$

$m = A \cdot s = 80\ m^2 \cdot 1{,}3\ t/m^3$

$= 104\ t/m$

$m = 250 \ldots 300\ t/shift$

$$v = \frac{m}{m} = \frac{250 \ldots 300\ t/shift}{104\ t/m}$$

= 2,4 ... 2,88 m/shift

corresponding to a drum depth of 625 mm: approx. 4 – 5 cuts

Production time t for 5 cuts with a shuffle time of 3 min per cut:

$$t = 5 \cdot \left(\frac{2 \cdot 40\ m}{3\ m/min} + 3\ min\right) = \mathbf{148\ min}$$

Possible output at a degree of utilization of $_t = 0{,}5$ and an actual shift time of $t = 6$ h:

$$m = {}_t \cdot t \cdot \frac{h_{FL} \cdot l_{St} \cdot T_w \cdot s_{FL}}{\frac{2 \cdot l_{St}}{v_{ESA}} + t_{shuffle}}$$

$= 0{,}5 \cdot 6\ h/shift \cdot 60\ min/h$

$$\cdot \frac{2\ m \cdot 40\ m \cdot 0{,}625\ m \cdot 1{,}3\ t/m^3}{\frac{2 \cdot 40\ m}{3\ m/min} + 3\ min}$$

= 394,4 t/shift

(≙ 3 production shifts per day ≈ 1200 t/d)

Number of personnel:

- machine operator, roof support operators 3 men/shift
- face/roadway junction, material handling 3 men/shift
- fitters, electricians, supervisors 3 men/shift

Total: 9 men/shift

Output per man and shift $\frac{400\ t/shift}{9\ MS/shift}$ **≈ 44 t/MS**